MINING engineering

MARCH 1954

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COVER

This month cover artist Herb McClure dramatizes the man with the Jackhammer, while within the magazine two authors discuss the more technical aspects of drill steel performance.

FEATURES

ARTICLES

TRANSACTIONS

—Personnel Service—

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18**; 100 Farnsworth Ave., **Detroit**; 57 Post St., **San Francisco**; 84 E. Randolph St., **Chicago 1**. Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Mining Engineer and Surveyor, 35, married. Experience: 2 years hard rock, 6½ years coal. Available immediately for Eastern, Central or Southern States for mining, sales or allied industry. M-60.

Mining Engineer, 36, 1940 Rolla B.S.E.M., Registered, married, family. Thirteen years coal mining, last 6 as chief engineer of large producer. To metallic and nonmetallic mining can offer broad experience in plant conveying and structural design, detail and construction. Domestic or foreign. Will travel. M-61.

Mining and Industrial Engineer. Many years experience exploration, drilling, testing, examination, development, and operation of open pit and underground mines, and development of natural resources, domestic and foreign. Inspection and report on natural resources for industries, metallic and nonmetallic. Available. M-62.

Mining Engineer, 34, single, M.S. degree, Registered P.E.; 13 years experience operation and management in hardrock and placer, includes 4 years mine examination, western U. S., Alaska, Canada. Will accept foreign location. Available short notice. M-63-4710-E-5-San Francisco.

Graduate Mining Engineer, 34, married, Canadian citizen, veteran. Six years experience gold, base metal, and industrial mineral mining. Experienced in development and production and Heavy-Media supervision. Seeks permanent position with future. M-64.

Mining Engineer, 48, graduate, very good health, married, one child. Twenty-six years experience as surveyor, superintendent, production engineer, supervisor, mines and quarries in gypsum, limestone, manganese, sulphides, iron, bauxite, and gold. Available short notice. M-65.

POSITIONS OPEN

Mechanical-Electrical Superintendent for large base metal operation in South Africa. Must be well versed in operation and maintenance, steam power plant, mine and mill plant treating 50,000 tons per month. Three-year contract. Transportation for self and family and salary while traveling. Climate and living conditions excellent. F9609.

Junior Mine Shift Bosses, college graduates, to work in underground mine operations. Salary to start, $4800 a year. Location, Chile. F9578.

Exploration and Development Engineers, 35 to 50, with considerable experience in examination and development of platinum deposits associated with laterites occurring in regions of ultra basic rocks. Knowledge of nature of occurrence of nickel silicates and cobalt in laterites weathered from ultra basic rocks also desired. Examination by trenching and auger drill. Analytical work in field. Development will involve layout of water service by dams, canals, and high lift pumping for ground sluicing of hillsides. Contract, 3 years. Salary, $7500 a year U. S. currency, plus allowances. Salary tax exempt. Location, Ethiopia. F9573.

COAL MINE SUPERINTENDENT—
age 50 married. 15 years as mining engineer, 13 years as superintendent. Experience with conveyors and all types of mechanical loading equipment. Have proven ability to oversee complete operation, including labor relations, preparation, maintenance of equipment, safety, and underground operations. Desires position as superintendent for progressive mining company.

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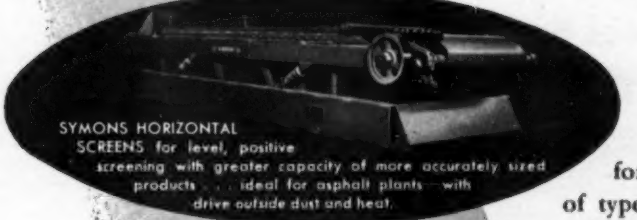

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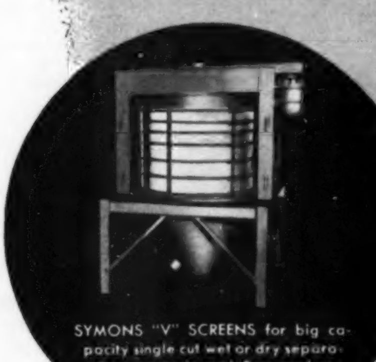

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S254

—Letters to the Editor—

Early Days in Iron

Referring to the article in MINING ENGINEERING, January 1954, *The Brown Iron Ore Resources of Missouri*, by E. L. Clark and G. A. Muilenburg, the following may be of passing interest to the readers.

Many years ago while acting as chemist for the old South St. Louis Blast Furnace I was allowed by the management to do outside work. A banker in St. Louis was operating a furnace in Dent County, the Cherry Valley furnace. I was to check samples of iron going to the South Chicago Steel Works; the iron was right on the Bessemer limit.

One night on deadman's shift, about 3 o'clock in the morning, there was a sudden electrical storm, a loud flash. The engineer was half asleep but the report woke him. The engine began to jump around, lightning struck the roof, ran down the exhaust pipe, burnt out the babbitt under the blowing engine. The engineer stopped the engine just in time.

An item told me by my banker was the following: There was a small furnace near Ashland, Ky., that ran on lump coal mixed with coke. A young man about 17 years of age who was helping in the office ran in very excited. He shouted, "We've broken the World's Record. We made 37 tons of iron yesterday."

Who do you suppose the young man was? It was William McKinley, later president of the United States.

Such was the romance of the early days in the iron business.

William W. Taylor
Signal Mountain, Tenn.

Cover Comment

For the past year I have taken great interest in the exceedingly smart covers on MINING ENGINEERING. They are far superior in design to the covers on most of today's publications. The artist has created excellent compositions of design and color in a direct relationship to the featured articles.

The recipients of MINING ENGINEERING should be proud of the unique, modern interpretations on its monthly publication.

Mrs. W. G. Wahl
York Mills, Ont.

These are kind words, indeed. The Editors shall certainly pass this comment from a member's wife along to the cover artist, Herb McClure.

Correction

The Personals item on Tom Lyon, MINING ENGINEERING, January 1954, p. 93, should be corrected to read as follows: He was assistant to the general manager (International Smelting & Refining Co.) from 1944 until his retirement in 1950.

WHAT FULL CONTROL MEANS

to the operator of an Allis-Chalmers TR-200 Motor Wagon

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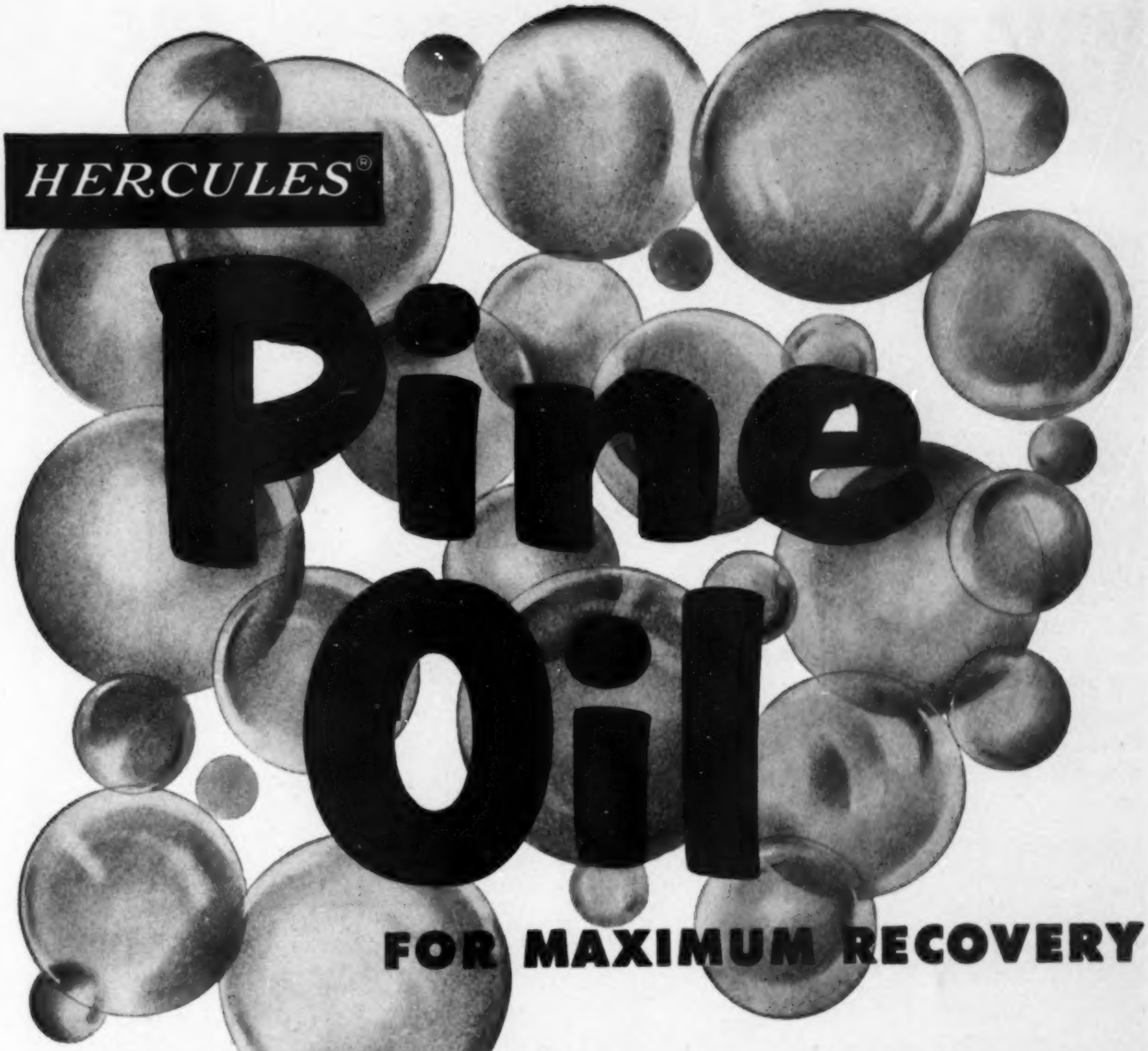

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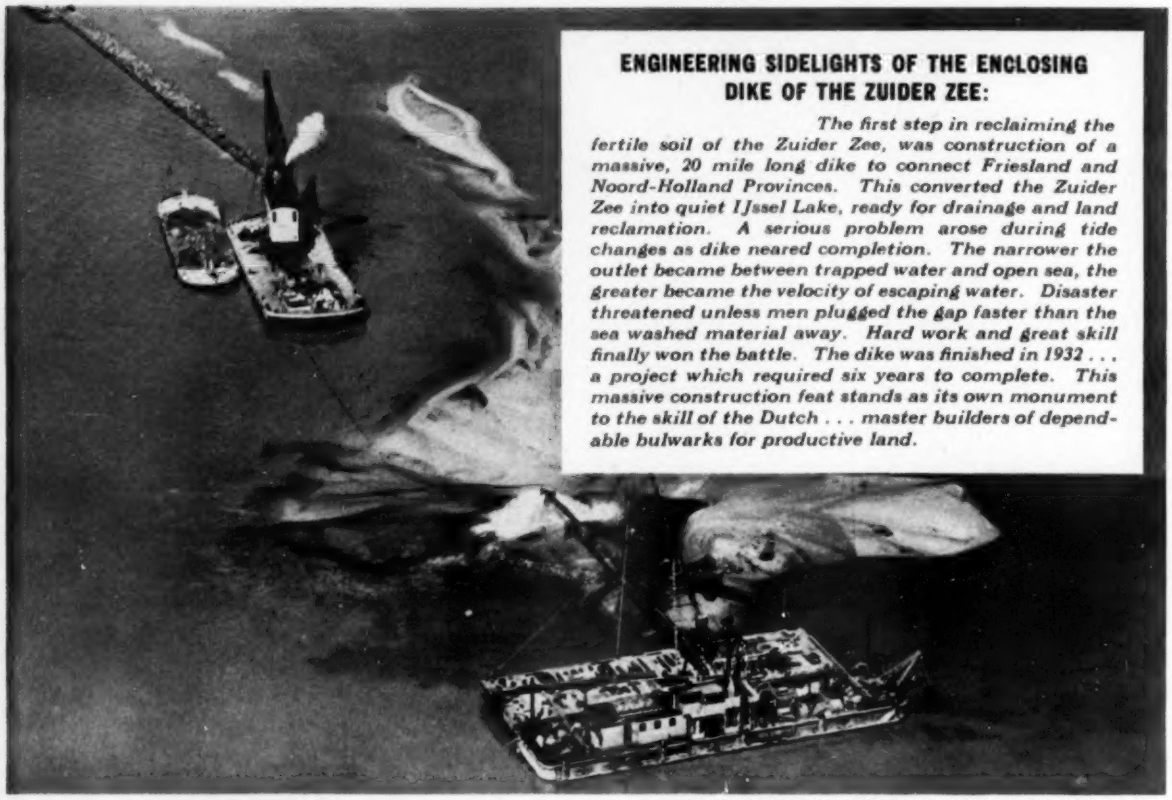

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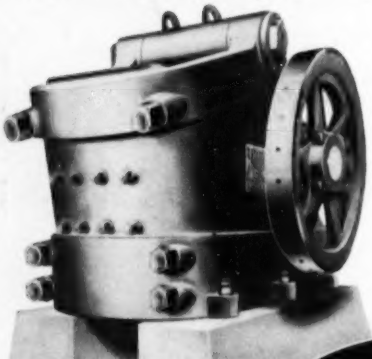

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Books for Engineers

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The Geology of the Rutland Area, Vermont, by William F. Brace, Vermont Geological Survey, *Vermont Development Commission,* East Hall, University of Vermont, Burlington, Vt., free, Bulletin No. 6, 120 pp., 1953.—Well-illustrated with photographs and diagrams. Three maps in pocket.

Publication List and **The Story of Ohio's Mineral Resources,** compiled by Dorothy G. Watkins and Ethel S. Dean, *Ohio Dept. of Natural Resources,* Orton Hall, Ohio State University, Columbus 10, Ohio, free, Ohio Geological Survey Information Circular No. 9, 47 pp., 1953.—Well-illustrated with maps and photographs of the industries developing and utilizing the various mineral resources of the State. The revised publication list contains reference to all reports and maps published by the Ohio Geological Survey.

Tulsa Geological Society, Volume XXI, available from E. T. Peterson, c/o Atlantic Refining Co., 830 Kennedy Bldg., Tulsa, Okla., $2.00, plus postage, 267 pp.—More than 25 articles appear in this 1953 Digest. Among them are an article on the Williston Basin, a review of development in the Denver-Julesburg Basin, and a report on the geology of the Uinta Basin. Three theses submitted to the University of Oklahoma are also included.

Dust Emissions from Small Spreader-Stoker-Fired Boilers, by Elmer J. Boer and Charles W. Porterfield, *Bituminous Coal Research Inc.,* 2609 First National Bank Bldg., Pittsburgh 22, Pa., 25¢, 8 pp.—A paper based on research sponsored by Bituminous Coal Research Inc., the national research agency of the bituminous coal industry. It was presented at the ASME 1953 spring meeting, Columbus, Ohio.

Water Pollution Abatement Manuals: *Manufacturing Chemists Assn.,* 1625 Eye Street, N.W., Washington 6, D. C.: **Organization and Method for Investigating Waste in Relation to Water Pollution,** Manual W-1, 20¢; **Insoluble and Undissolved Substances,** Manual W-2, 20¢; **Neutralization of Acidic and Alkaline Plant Effluents,** Manual W-3, 25¢.

Geologic Map of East Tennessee with Explanatory Text, compiled by John Rodgers, *Dept. of Conservation, Div. of Geology,* Nashville, Tenn., Bulletin 58, Part II, $3.50, 168 pp., with plates, figures, tables, 1953.—The accompanying text describes the rock units shown on this new geologic map of the Unaka Mountains and the Valley of East Tennessee. The only previously existing geologic map of this region was one prepared between 1890 and 1907.

Tables of 10^x (Applied Mathematics Series No. 27) *National Bureau of Standards,* Available from Superintendent of Documents, Government Printing Office, Washington 25, D. C., $3.50 543 pp., 1953.—Antilogarithms to the base 10 are tabulated to 10 decimal places from 0 to 1 at intervals of .00001; a revision, with corrections, of the Dodson table of 1742.

Tables of Natural Logarithms for Arguments Between Zero and Five to Sixteen Decimal Places (Applied Mathematics Series, No. 31) *National Bureau of Standards,* Available from Superintendent of Documents, Government Printing Office, Washington 25, D. C., $3.25, 501 pp., 1953.—This table of logarithms to the base e covers the indicated range of intervals of .0001. As in the case with all volumes in this series an introductory section explains both the methods of calculation and the effective use of the tables.

Silicified Middle Ordovician Trilobites, by H. B. Whittington and W. R. Evitt, II, (Memoir 59), *Geological Society of America,* 419 West 117 St., New York 27, N. Y. $3.00.

Design for Production, Anglo-American Council on Productivity, Report PB 106482, available from *Office of Technical Services, U. S. Dept. of Commerce,* Washington 25, D. C., $.90, 81 pp., photographs, charts, diagrams, tables.—The importance of careful product design in the U. S. in ensuring economical manufacture and the maximum use of readily available standard parts is described in this report by a British productivity team which visited the U. S.

The Canadian Mineral Industry in 1951, *Mines Branch, Dept. of Mines & Technical Surveys,* Ottawa, Canada, 50¢ Canadian, No. 841, 170 pp.—This technical report is divided into three sections, metals, industrial minerals, and fuels. The introduction covers the principal developments in the mineral industry as a whole for the year.

Effect of Germanium on the Transformation of White to Grey Tin at Comparatively Low Temperature, by R. R. Rogers and J. F. Fydell, *Mines Branch, Dept. of Mines & Technical Surveys,* Ottawa, Canada, 25¢ Can., Technical Paper No. 5, 11 pp., 1953.

The Determination of Uranium in Concentrates by the Fluorophotometric Method, by J. B. Zimmerman, F. T. Rabbitts, and E. D. Kornelsen, *Mines Branch, Dept. of Mines & Technical Surveys,* Ottawa, Canada, 25¢ Can., Technical Paper No. 6, 9 pp., 1953.

ORDER YOUR BOOKS THROUGH AIME—Address Irene K. Sharp, Book Department. Ten per cent discount given whenever possible. Order Government publications direct from the agency concerned.

The Economic Almanac 1953-1954, edited by Frederick W. Jones, compiled by the National Industrial Conference Board, *Thomas Y. Crowell Co.,* $3.95, 740 pp., 12th edition.—Founded in 1916, the Conference Board is an independent, nonprofit institution for business and industrial research. More than 3000 subscribing associates made the work of this organization possible. This is the first year that this handbook of useful facts about business, labor, and Government in the U. S. and other areas has been made available to those outside NICB membership.

Dynamic Equipment Policy, by George Terborgh, $4.50, 290 pp. **MAPI Replacement Manual,** $4.00, 75 pp. **Company Procedure Manual on Equipment Analysis,** $5.00, 45 pp. Three-volume sets of the above, $11.50, *Machinery & Allied Products Institute,* 120 South LaSalle St., Chicago 3.—The Machinery & Allied Products Institute is a federation of trade associations, manufacturing companies, and subscribing associates established to bring the diverse industries manufacturing the many kinds of industrial equipment and capital goods into a unified and homogeneous organization. These three volumes are designed to answer these questions: What equipment in your plant is economically replaceable? When will particular equipment be economically replaceable? What is the cost of not replacing?

Industrial Inorganic Analysis, by Roland S. Young, John Wiley & Sons, $5.75, 368 pp., 1953.—The author, who is with INCO, New York, has provided 31 chapters on individual metals or groups of metals. Dr. Young covers theory and practice, analytical procedures, necessary separations, manipulative technique, and evaluation methods for universal and specialized work.

Field Conference Guidebooks of the New Mexico Geological Society, *New Mexico Bureau of Mines and Mineral Resources, Socorro, N. M.* **2nd Field Conference, South and West Sides of the San Juan Basin, New Mexico and Arizona,** $5.00, 167 pp., 69 illustrations, October 1951. **3rd Field Conference, Rio Grande Country, Central New Mexico,** $5.00, 126 pp., 51 illustrations, October 1952. **4th Field Conference, Southwestern New Mexico,** $5.00, 153 pp., 69 illustrations, October 1953.

Chemical Analytical Methods, A handbook of Colorimetric Chemical Analytical Methods (for Industrial, Research, and Clinical Laboratories), *Tintometer Ltd.*, England, U. S. distributors: Messrs. Curry & Paxton Inc., $5.00, 206 pp., 27 illustrations.—A collection of various detailed technical instructions prepared for the use of laboratory technicians who employ the Lovibound Comparator and the B.D.H. Lovibond Nessleriser. Each method is the result of collaboration between Tintometer Ltd. and one or more leading laboratories, chosen as being the recognized authority in that particular test.

Safety Code, *Prevention of Accidents Committee of the Transvaal Chamber of Mines,* Johannesburg, South Africa.—In 1953 the Prevention of Accidents Committee celebrated the 40th anniversary of its campaign against mining accidents. This book, its most recent publication, is organized into 11 sections and contains 59 illustrations. It is a compilation of reports and recommendations received from numerous and varied sources concerning methods and devices for attaining mining safety.

Silicones and Their Uses, by Rob Roy McGregor, *McGraw-Hill Book Co. Inc.*, $6.00, 302 pp., 1954.—A manual in nonchemical language. This book gives a history of early investigations and the commercial development of silicones; describes the physical properties and applications of fluids, compounds, lubricants, resins, rubber, and other silicone products; and tabulates applications made by representative industries. There are also chapters on physiological response to silicones and on the preparation of silicones.

Geology of Eel River Valley Area, Humboldt County, California, by Burdette A. Ogle, *California Dept. of Natural Resources,* $2.75, 128 pp., 1953.—Bulletin 164 was prepared in fulfillment of the requirements for the doctorate at the University of California. Photographs, maps in pocket.

Geology of the Breckenridge Mountain Quadrangle, by T. W. Dibblee, Jr., and Charles W. Chesterman, *California Dept. of Natural Resources,* $1.75, 56 pp., 1953.—Bulletin 168 has photographs and three maps in pocket.

Chemical Process Machinery, by E. Raymond Riegel, *Reinhold Publishing Corp.*, $12.50, 750 pp., 2nd ed., 1953.—This new and greatly enlarged edition lists only equipment now commercially available. Diagrams, photographs, and tables illustrate the construction and operation of recently employed devices as multispheres, multicylinders, and high-speed centrifugal separators.

SANTA RITA, N. M., with its 37 families is being picked up and moved 1½ miles north by Kennecott Copper Corp. The town, owned by Kennecott Copper Corp., is moving to make room for continued expansion of the open pit operation. It will cost Kennecott about $1.5 million to do the job.

Only one of its kind in Canada, a tonnage oxygen unit has been installed at International Nickel Co.'s Copper Cliff plant in connection with its flash smelting process for copper. Turning out 300 tpd, the unit eliminates the need for other fuels in the process. It was built by Canadian Liquid Air Co.

A REYNOLDS MINES survey team has investigated the Cockpit country of Jamaica, and analysis of bauxite indicates that quantities exist equal to any of the current operations on the island. It is reported that negotiations will be opened with the Jamaican Government for mineral rights.

Colorado School of Mines and Harvard Business School have established a fellowship designed to produce leaders in the minerals industry. A Mines graduate who is awarded the fellowship will be enrolled annually for a 2-year course in business administration at Harvard. Financial support for the fellowship comes from 12 executives connected with the mining field.

Lehigh Coal & Navigation Co. expects to start mining uranium near Mauch Chunk, Carbon County, Pa., this spring. Mining operations on the recently discovered deposit will be carried on in a "very small way" at first, according to Robert V. White, Lehigh president.

M. A. HANNA COAL & ORE CO. sold its sinter plant and other machinery at the Clifton mines south of DeGrasse, N. Y., to U. S. Steel Corp. The sinter plant will be dismantled and shipped to Ohio for U. S. Steel's Cleveland mill.

SHIPMENTS OF CONCENTRATES from the new Sherritt Gordon mine at Lynn Lake, Manitoba, are being stockpiled at Fort Saskatchewan in preparation for start of operations at the new processing plant.

What appears to be the first real drop in price for titanium metal mill products was announced by Titanium Metals Corp. Reductions will average more than 12 pct for commercially pure sheet and plate. Price cuts are confined to charges added to base prices for certain manufacturing operations to meet needs of individual buyers. Base price of $15 per lb for sheet and $12 per lb for plate remain unchanged.

PHELPS DODGE CORP. is cutting refined copper production by about 1000 tons per month by reducing operations at the Morenci and Ajo pits in Arizona from 6½ to 6 days per week. Production at the two mines has been about 15,900 tons per month.

Certificate of necessity for $55.25 million was issued to Nicaro Nickel Co., subsidiary of Freeport Sulphur Co., for development of facilities at Moa Bay, Cuba, and New Orleans. It will allow the company to write off 80 pct of estimated cost in 5 years, for Federal tax purposes. About $18 million will be spent for a pilot nickel processing plant near New Orleans, $35 million for mining development at Moa Bay, and remainder for ore carriers. Office of Defense Mobilization also announced that National Lead Co. has been granted a $43 million allocation for 75 pct expansion of the Government-owned Nicaro Nickel plant.

Texas City Chemicals Inc. is operating a production unit for recovery of uranium concentrate as a byproduct at its new phosphate plant at Texas City, Texas. The unit is designed to recover uranium concentrates as a byproduct of feed and fertilizer grades of dicalcium phosphate from Florida rock having minor amounts of uranium.

MANAGEMENT OF PATINO MINES & ENTERPRISES has just about given up hope that it might resume operation of its Bolivian tin mines. It was revealed at a recent stockholders meeting that instead, the company hopes to start negotiations for a fair valuation and method of payment with Bolivia.

A bill introduced into the 39th Colorado General Assembly by a group of State Senators reads: "Section 19: The General Assembly shall have the power to levy taxes on irreplaceable natural resources separated from the earth or waters of the State. All Revenues derived from this source shall be a part of the general revenues of the State."

Manufacturers News

Remote Control Jumbo

One operator can move into position with the new *Gardner-Denver* JSP Mobiljumbo, and drill a complete round, without leaving his seat on the carriage. Positioning of booms and drills, as well as the complete drilling cycle, are hydraulically controlled on the JSP, designed to reduce time, effort, and cost of drilling.

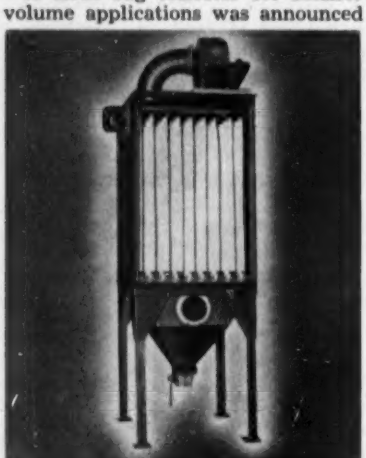
One lever raises the machine on jacks for stability while drilling. Drills can be positioned for down or up holes, as well as horizontal, adapting unit for also drilling roof bolt holes. Complete drilling cycle, including steel speed, feed pressure, hole cleaning, and steel withdrawal, is easily controlled from operator's seat. Boom lengths are available for handling faces up to 14 ft high, and face widths to 25 ft can be drilled off a single setting of the Mobiljumbo. **Circle No. 1**

Transportometer

Sintering Machinery Corp. has designed an offset Transportometer weighing scale for belt conveyor installation with limited overhead clearance. Weight integrator scale levers and suspension are located under conveyor, integrator can be located at either side. **Circle No. 2**

Dust Collector

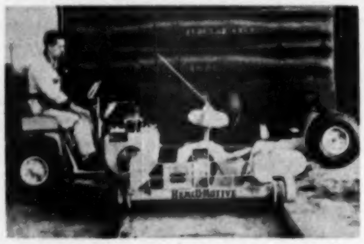
A cloth bag collector for smaller volume applications was announced



by *Pangborn Corp.* Type CN self-contained units are shipped ready for installation in seven sizes, from 200 to 1000 sq ft area. **Circle No. 3**

Switching Cars

Aimed to slash car handling costs, *Hemco-Motive* handles up to 3 rail cars, operates on rails or over the



ground. This car switcher is gasoline powered for instant starting, and with overall width of 41 in. it goes almost anywhere. **Circle No. 4**

Bearing Seals

Hycar rubber bearing seals to keep lubrication in and dirt out are said to help maintain long life and high efficiency of belt conveyor

idlers. *B. F. Goodrich Co.* claims product to be tough, resilient, and impervious to oils, grease, and most chemicals. **Circle No. 5**

Air Classifier

Hardinge Co. has a new dry classifier for continuous separations of coarse and fine airborne particles. It can be used for closed circuit with pulverizing mills and as a self-contained unit. Raw mixture of coarse and fine particles feeds from bottom, oversize not removed by blade impact drops out in eddy current above rotor and is deposited on outer shell by centrifugal action. Returning oversize is cleaned of fines by winnowing action of rotor blades. **Circle No. 6**

V-Belts

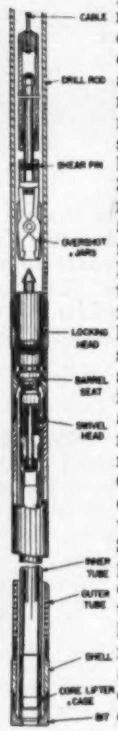
Easy coupling, greater strength, longer life, and perfect balance are claims for Veelos adjustable V-belts available from *Manheim Mfg. & Belting Co.* **Circle No. 7**

Plastic Pumps

Jabsco Pump Co. has a series of low cost, self-priming, plastic pumps delivering up to 350 gph. The small, compact, and long-lived pumps should find wide metallurgical use handling flotation reagents and corrosive materials. **Circle No. 8**

Fast Core Drilling

Wire line core barrel equipment, recently introduced by *E. J. Longyear Co.*, promises to revolutionize exploratory drilling, according to the company. The small diameter wire line equipment, under development since 1947, has been successfully used in BX holes of depths of 500 to 3200 ft, and a hole is currently being drilled to 5000 ft with this equipment. Principle of wire line core barrel, used by the petroleum industry for large diameter holes, has never before been successfully applied to small diameter core drilling. Significant features are: (1) The string of rods is pulled only when replacing the bit. (2) Core is extracted at the end of each run with a wire line cable reel. Skilled operators can pull core, and return empty inner tube in 15 to 20 min with a 1500-ft hole. Round-trip time for a 3000-ft hole can be as low as 32 to 38 min, contrasted for 3 to 4 hr commonly required with conventional equipment at this depth. (3) The core barrel has ball bearing, swivel type head, with optional water shut-off valve. Latter device prevents circulation of fluid when core block occurs, and alerts operator to pull core before grinding starts. **Circle No. 9**



Gas Detection

Announced by *Mine Safety Appliances Co.* as an exceptionally accurate, portable instrument for SO_2 detection, new unit has range for fast detection of 0 to 50 ppm in working atmospheres. An aspirator bulb is connected to a tube that has a graduated scale. The reagent in the tube turns from blue to white, with length of decolorization proportional to percent SO_2 in the sampled air. **Circle No. 10**

Moisture Meter

Said to be simple and easy to operate, the portable Olivo moisture meter determines surface moisture content of material in less than 2 min, with accuracy of 2 pct, when properly calibrated. Meter is available in U. S. exclusively through *Heyl & Patterson Inc.* **Circle No. 11**

Free Literature

(21) UNDERGROUND DRILLING: Bulletin No. 340 from *Sprague & Henwood Inc.* illustrates and gives complete working data for two sizes of air-operated machines for either diamond core drilling or blast-hole drilling underground. A rod-pulling device, with air-actuated piston, is described for each machine. The bulletin also summarizes S&H's wide line of diamond bits and other accessory equipment for diamond core drilling of any kind.

(22) SLIDE CHART: *Lebanon Steel Foundry* has a slide chart on steel casting material selection with data on 36 grades in production at Lebanon Steel. Guide arrow reveals nominal analyses, minimum mechanical properties, and heat treatment of the grade. The chart also gives the comparable Government and industry designations where applicable. A special section provides reference facts for eight different heat resistant cast steel alloys.

(23) HARDFACING ALLOYS: *Amsco's* hardfacing and buildup rods are fully described in a 48-page, 2-color catalog. Typical applications are shown in over 70 photographs and full metallurgical and physical properties are included for the 4 automatic and 15 manual-use rods and electrodes.

(24) FLEXIBLE TUBING: Application of flexible metal tubing for air, oil, steam, gases, and volatiles is explained in an illustrated book from *Pennsylvania Flexible Metallic Tubing Co.* Pictured are various Penflex products: steel, bronze, and aluminum tubing and hose in many types of construction; blower and ventilation hose, tar and asphalt hose; diesel piping and barrel fillers.

(25) COLLOID: *Stein, Hall & Co. Inc.* has free testing samples and a booklet on Jaguar, a natural colloid derived from guar seeds. Used in mineral dressing as a conditioning reagent, depressant, and flocculant, Jaguar is said to be efficient and economical per unit cost.

(26) VERSATILE DIESELS: "Cummins Cuts Contractors' Costs" is an illustrated booklet from *Cummins Engine Co. Inc.* Cummins diesels are

shown helping construct the New York State Thruway, dredging river bottoms, crushing rocks, pushing cargo boats, lifting concrete.

(27) SYNCHRONOUS MOTORS: Standard construction features of engine type synchronous motors in ratings of 100 hp and larger for speeds of 450 rpm or less are described in a bulletin from *Allis-Chalmers Mfg. Co.* While open-type construction is generally used for engine-type motor applications, A-C points out that such protection as drip-proof or drip-proof protected construction can be provided as well as enclosed or explosion-proof collector assembly.

(28) pH METER: No line-operated pH meter has ever been offered for so low a price, says *Photovolt Corp.* Bulletin No. 225 emphasizes that the simplicity and low price of model 15 have been achieved by taking advantage of the most recent advances in electronic tubes and circuits. Accuracy and stability are not sacrificed.

(29) AIR CONDITIONER: Bulletin No. 1307 from the *Dravo Corp.* describes a Pulpit air conditioner designed and developed for use in industrial control areas where heat, dirt, or fumes create hazardous working conditions.

(30) Cu-Mn-Sn: Originally published in the Journal of the Institute of Metals, "The Structure and Mechanical Properties of Copper-Manganese-Tin Alloys," has been reprinted by the *Tin Research Institute.* This paper describes a range of new alloys that may be useful in the fields formerly served by nickel silver.

(31) MILLMAN'S SLIPSTICK: *Denver Equipment Co.'s* new mill bulletin has a slide rule on the back cover to determine mill size or capacity. In addition there are drawings, flowsheet data, and specifications on Denver's Steel-Head Ball-Rod mills. Capital cost and power consumption figures are provided in a comparison of one vs two-stage grinding. Power consumption is indicated for high speed conventional rod mills.

(32) STARTERS & PUSH BUTTONS: Selection information on motor starters and push buttons is presented in GEA-6061 from *General Electric Co.* In addition to rating and suggested prices, the folder contains photographs of the equipment, circuit diagrams, brief application data.

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

Not good after June 15, 1954 — if mailed in U. S. or Canada

Please send me { More Information ☐ / Free Literature ☐ / Price Data ☐ } on items circled.

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11	12	13	14	15	16	17	18	19	20
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61	Students are requested to write direct to the manufacturer.								

Name ____________________ Title ________

Company ____________________

Street ____________________

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(33) LIGHT WEIGHT PIPE: Typical applications of *Naylor Pipe Co.* light weight lockseam-spiralweld pipe and fittings are shown in bulletin No. 507. Included are standard specifications on pipe from 4 to 30 in. diam, together with data on fabricated fittings, flanges, and connections to meet all pipe line requirements.

(34) V-BELT: *Raybestos-Manhattan Inc.* has a V-belt which, the company says, has horsepower capacity 40 pct above standard V-belts. Descriptive folder 6628 gives further data on this oilproof, nonspark, heat-resistant, all synthetic rubber belt.

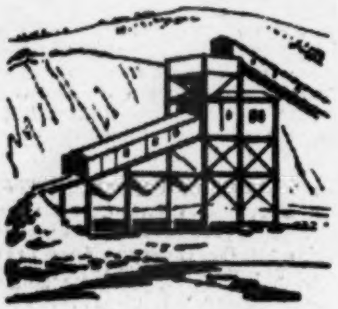
(35) PIT & QUARRY $$$: "Profit Builders in Pit and Quarry" has photographs of the complete line of *Caterpillar* equipment available to pit and quarry companies. Cat diesels, track-type tractor shovels, bulldozers, rubber-tired earthmovers, motor graders, and four-wheel scrapers are shown in actual use in all parts of the U. S.

(36) WELDING DATA: *Wall Colmonoy Corp.* has a 3-page engineering data sheet describing Walmang, austenitic nickel manganese steel, bare and coated, welding electrodes that produce a weld deposit having the general characteristics of 12 pct manganese steel.

(37) DUST COLLECTOR: *Pangborn Corp.'s* bulletin describes the unit type "CN" dust collector made to meet the need for an economical, yet highly efficient dust collector for smaller volume applications. Designed for indoor use, the collector permits clean air to be discharged inside the plant, a distinct savings where air is heated or cooled.

(38) MATERIAL HANDLING: *Stephens-Adamson Mfg. Co.* has two new bulletins; one covers car pullers, the other is on hand and motor winches.

(39) STOCKPILE LOADING: A booklet from *Athey Products Corp.* shows the Athey Force-Feed HiLoader on a variety of applications, gives specifications and application data, and details the many features that give "high capacity loading for high profits."



(40) TRUCKS: Introduced in *Automatic Transportation Co.'s* 4-page catalog of electric-driven industrial trucks is the new Dynamotive, the first gas fork lift truck with electric transmission. The Dynamotive is said to combine the long service and trouble-free operation of straight electric trucks, with the constant power source of a gas engine.

(41) SOIL SAMPLING: The science of soil mechanics has become one of the important factors in foundation engineering and design. *Acker Drill Co.'s* bulletin 25 illustrates and describes soil sampling tools in general use and shows Acker's complete line of soil sampling tools, diamond and short core drills, drilling accessories and equipment.

(42) ROOF BOLTING: Mining Products Div., *Equipment Corp. of America* has improved Roof-Lock steel expansion anchor. The Plug-Lock model is claimed to hold the plug in position and prevent the anchor from dropping as the bolt assembly is installed and tightened.

(43) DEWATERING FILTER: *Hardinge Co.* expects the Tray Belt filter, a "new and radically designed" dewatering filter, to have many applications for continuous liquid-solids separation in the mining, chemical, and industrial fields. Bulletin No. AH-450 presents operating details, applications, and operational advantages over conventional types.

(44) GATES & VALVES: A bulletin from *Stephens-Adamson Mfg. Co.* covers slide gates, quadrant gates, and valves. Unusual units include the Twistite double closure bin valve and the Moore bin gate. The former unit consists of two rubber sleeves which are twisted shut by a rotating collar, forming a dust and watertight lock. The Moore bin gate, for heavy duty applications, consists of an apron belt which can be easily rolled away from the bin opening.

(45) REFRACTORIES: Two bulletins from *Ironton Fire Brick Co.* are of interest to industries where refractories are used. Bulletin 103 covers specific applications of Ironton Steel high duty dry press fire brick. Bulletin 104 presents ten new refractory insulating concretes that are hydraulic setting mixes of aluminum silicate base for service temperatures up to 3000°F.

(46) BUILT TO TAKE IT: *Allis-Chalmers Mfg. Co.'s* earth moving equipment is pictorially described in "Presenting the Allis-Chalmers Line." The catalog has been expanded to 36 pages to include data on the HD-15 crawler tractor now available with a choice of two drives—standard transmission or hydraulic torque converter.

(47) DOCKS, WHARVES, PIERS: *DeLong Corp.'s* 12-page illustrated booklet describes new technique of dock construction which eliminates 90 pct of field labor in the erection of docks, wharves, and piers—temporary or permanent. These steel docks can be prefabricated in any shipyard where costs are controlled. Then the DeLong dock-barge, carrying its own installation gear, is towed to site. In a few days, barge is transformed into a dock, ready to berth ocean-going ships.

(48) DETECTOR: Presented in a 16-page booklet from *Fenwal Inc.* are application ideas, installation tips, and suggested circuits for utilizing Detect-A-Flo, a device that controls or detects both liquid level and air flow. Detect-A-Flow has no moving parts, is hermetically sealed and installed directly in the tank or air stream being monitored.

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Climax Rates as Largest Underground Operation in U. S.

Climax Molybdenum Co.'s mine at Climax, Colo., is the largest underground mine in North America. No underground mine on the continent has ever reached the 27,000 tpd mark set in January by the molybdenum mine, according to Arthur H. Bunker, company president.

Aside from its reputation as one of the most difficult words in the English language, molybdenum is used chiefly to harden and toughen automotive steels and cast iron. It is an essential alloy for hardening armor plate and gun barrels, and for resisting high jet engine temperatures. Molybdenum sulphide has unique properties as a lubricant.

In announcing the rate of mine output, Mr. Bunker also stated that "Three years of world scarcity are now a thing of the past. As a result of the expansion at Climax and the consequent lifting of export quotas by the Federal Government, every nation in the free world will henceforth be able to get all the molybdenum it needs. For national security reasons, export licenses will still have to be obtained for each shipment."

Climax has been noted in the past for having the highest year-round post office in the U. S. And with the installation of television antennas at 13,770 ft atop nearby McNamee Peak and the wiring of the town's 500 homes, Climax became the world's highest television community. The mine is the second largest underground installation in the world, second only to the Kennecott Copper Corp.'s Braden mine in Chile.

Mining at Climax is carried on by block caving, collapsing a mountain from the inside. More than 8 million tons of ore will be extracted during 1954. This year, according to the company, it will supply about two thirds of the metal produced this side of the Iron Curtain. Published sources rate Soviet production at less than one fortieth of that in the Free World. World production in 1954 is expected to be more than 65 million lb.

The recent expansion program was entirely company financed. Climax increased mine and mill capacity 78 pct from about 24 million lb of molybdenum in 1952 to an anticipated 42.5 million lb in 1954, including the 5 to 8 million lb new capacity added most recently.

The "glory hole" at Climax Molybdenum mine can be seen in the center. Climax has the highest year-round post office in the U. S., and is the highest television community in the world. Bordered by the Continental Divide, the town is 11,400 ft high.

Creighton Increases Concentrator Capacity

Capacity of the concentrator at International Nickel Co.'s Creighton mine, Copper Cliff, Ont., has been expanded to 12,000 tons of ore per day, according to J. Roy Gordon, vice president and general manager of Canadian operations.

Originally designed for a capacity of 6000 tons of nickel bearing ore in 1948, outbreak of Korean hostilities in 1950 brought an increase to 10,000 tons in planned capacity. Currently, the mill building has been enlarged to a length of 465 ft and is 175 ft wide and 70 ft high. Two crushers have been added to the original four, and the 36 added flotation machines bring the total to 180. INCO is producing refined nickel at an annual rate of about 275 million lb.

The concentrator is at the site of the Creighton No. 7 shaft, with headframe and hoist house integral parts of the mill building. Ore from the mine is hoisted directly into the crushing plant. Part of the mill feed is carried by conveyor from another Creighton shaft two thirds of a mile away.

Plant water supply is obtained from a 6-mile pipeline and bulk concentrate is pumped through another pipeline to INCO's reduction plants at Copper Cliff, 7½ miles away.

Kennecott Curtails Chino, Ray, Operations

The "marked decline in the demand for copper" has forced Kennecott Copper Corp. to curtail operations at the Ray Mines Div., Ray, Ariz., and the Chino Mines Div., Hurley, N. M.

Beginning March 7, Ray goes on a 5-day week operating schedule. March 8 Chino follows. Approximately 300 men will be laid off on the basis of seniority at Chino, while another 55 will be laid off at Ray under the same system. Recalls will be on the same basis. Operations on Saturday and Sunday have been suspended at both mines.

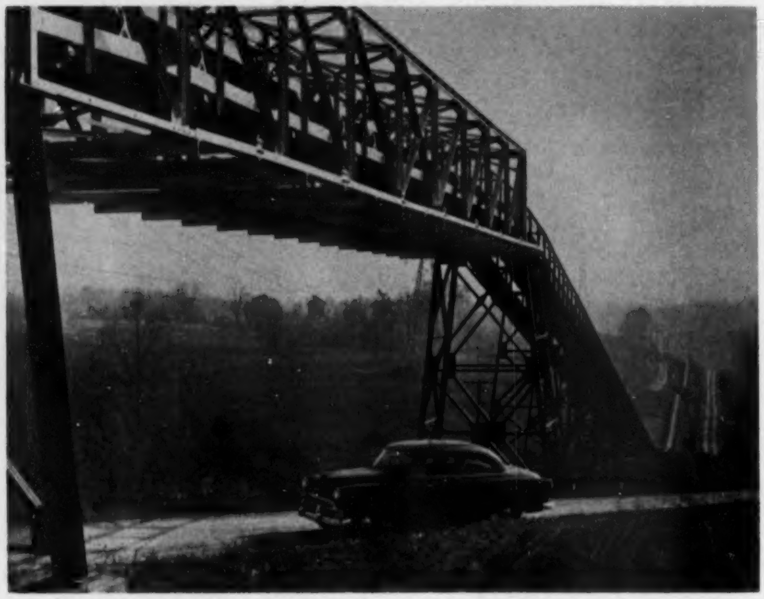
Beltroad Delivers Coal Over 4½ Mile Obstacle Course

One of the longest permanent rubber conveyor belt systems in the world is carrying coal from a southern Ohio strip mine near Beverly to an Ohio Power Co. storage area on the banks of the Muskingum River. The storage area serves Ohio Power's new 400,000-kw plant. The conveyor system is located on an isolated, wooded, rolling tract 5 miles northwest of Beverly. The belt transports 800 tons of coal per hr over a 4½ mile, twisting, up-and-down hill course. The belt hurdles country roads, spans a state highway, and crosses the 500-ft wide Muskingum.

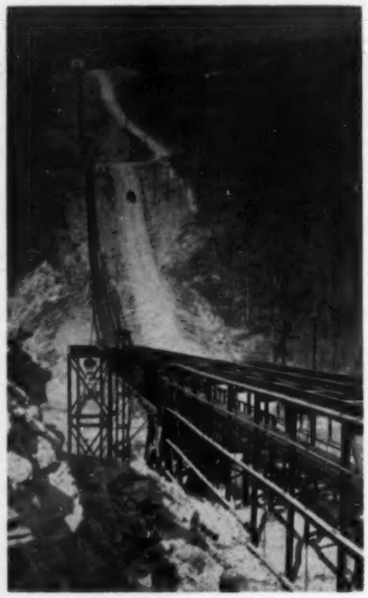
Ohio Power has coal reserves which can feed the new conveyor system for a number of years. B. F. Goodrich Co., Akron, Ohio, built the conveyor system. It consists of 14 flights, or sections, of rubber conveyor belting, ranging in length from 500 to 2964 ft, pulley to pulley distance. Belts are 36 in. wide and travel at 600 fpm. Highest incline angle traversed is 12°; greatest decline is 12°. Coal discharges from one belt to another automatically at transfer points.

Mined coal is loaded by electric shovel on Euclids for transportation to a preparation plant for washing and crushing. It then starts its roller-coaster ride to the stockpile.

The 14 conveyor drive motors are capable of producing a total 1435 hp. To install the 48,000 ft of rubber belting needed for the beltroad, 50 rolls of belting, weighing up to 4½ tons each, were hauled to the site by motor freight from the Akron plant. A portable electric vulcanizer, weighing more than a ton, was used to make the vulcanized splices which make the belts endless.



The beltroad bridges all sorts of handicaps. Here it crosses a country road and descends the rolling hillside below. From this point, astride what was once a cornfield, the beltroad still has 1½ miles more to go to the Muskingum River.



Foreground section of belt climbs at an angle of 12°. Auto in background will have to shift to low to climb hill.

Expand Unloading Dock At Sparrows Point Plant

Recent 1000-ft extension of its ore dock basin to 2200 ft has made for greater ore loading facility at the Sparrows Point, Md., plant of Bethlehem Steel Co. The plant normally receives 7 million tons of ore annually.

Three ocean-going vessels can now be unloaded simultaneously, while a fourth vessel is preparing to leave after discharging its cargo.

The extension project necessitated dredging to 40-ft mean-low-water and a removal of 1.35 million cu yd of material. Other facilities installed were a 15-ton ore unloader and a 20-ton combination ore unloader and bridge. The added equipment permits a greater unloading capacity of 2880 tph.

Three ocean-going vessels unload, while a fourth prepares to leave the newly enlarged ore dock at Bethlehem Steel Co.'s Sparrows Point, Md., plant.

Monazite Sands Are Spurring Idaho Dredging Operations

The Bret Harte type of character that once typified Idaho mining is rapidly vanishing before the onslaught of dredges, logging camps, and crawler tractors. Creeks that were once sluiced for gold are now being dredged for monazite sands—which someday may lead in value all other Idaho minerals.

Monazite contains thorium, an atomic material, and its production is under the control of the Atomic Energy Commission. While the Government buys some of the monazite, its use is top secret. Air Force Secretary Harold Talbott wants Congress to authorize subsidies for its production. Current domestic production is only about 2800 tons per year. The goal of 25,000 tons by 1956 is said to be short of what will be needed.

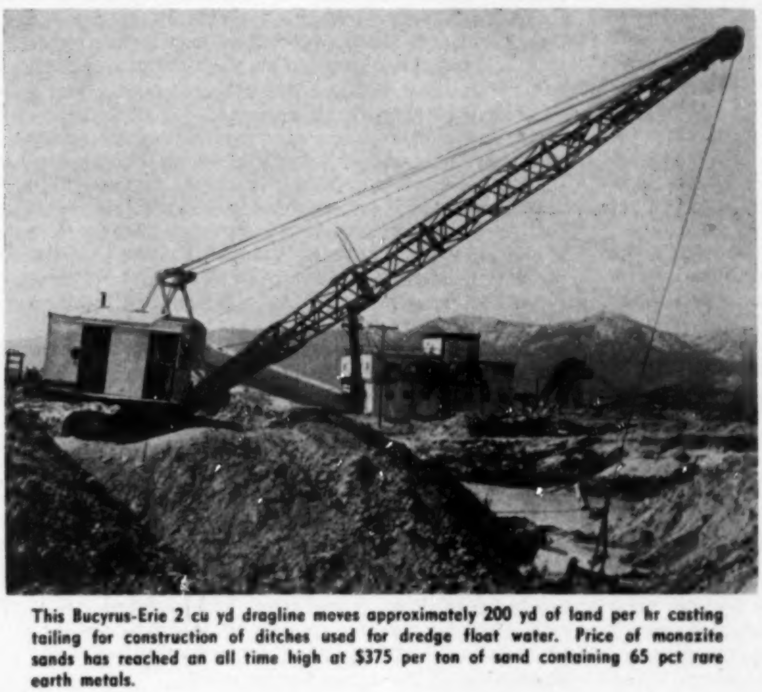
Until about 1909, significant amounts of monazite were dredged in the Carolinas and Florida. From 1909 to 1949, the U. S. imported most of its monazite sands. About 75 pct of it came from Travancore, India, and the rest from the states of Espirito Santo, Bahia, and Rio de Janeiro in Brazil. A hectic search for other sources started when India clamped an embargo on monazite sands. Idaho is one result of the search.

Monazite contributes thorium nitrate for thoriated filaments in electronic tubes. Television studio lights contain rare earth oxides and fluorides in the carbon cores. In high grade steel production rare earth metals improve rolling qualities. During World War II, the Germans found that alloying 0.3 pct misch metal with magnesium improved forging and heat-resistant qualities of airplane parts. Zircon is used in ceramics and refractories and zirconium is employed for surgery in bone splicing.

Equipped with a bulldozer blade, a Caterpillar tractor clears an area for the dredge operation. Frequently, the tractor is called upon to pull trucks out after they become mired in the swampy ground.

Price is higher than ever before at about $375 per ton of sands containing 65 pct rare earth metals. A ton brought $180 in 1922. The price slumped later, and even after World War II only reached $60 per ton.

H. E. Kremers, research director for Lindsay Chemical Co., sifts the fine golden brown sand that miners once cursed because it clogged sluice box riffles. Lindsay processes about 85 pct of monazite sand mined in U. S.



This Bucyrus-Erie 2 cu yd dragline moves approximately 200 yd of land per hr casting tailing for construction of ditches used for dredge float water. Price of monazite sands has reached an all time high at $375 per ton of sand containing 65 pct rare earth metals.

Three 1000-ton dredges are operating in the mile-high Cascade Basin. A logging operation is running ahead of the dredges, taking out some 4 million board ft annually in clearing the land. J. I. Morgan Inc., New Meadows, Idaho, is the logger.

The dredges, electrically powered, are being operated by Baumhoff-Marshall Co., Cascade, Idaho; Warren Dredging Co., Boise; and the Idaho-Canadian Dredging Co., Cascade. The dredges handle from 4000 to 6000 cu yd of gravel per day on a 24 hr basis. At the present rate, monazite deposits in Idaho promise operation for 25 years or more.

TWO things figure in the continuation of the Lake Superior region as a mighty industrial area, according to E. M. Richards, vice president in charge of planning and development, Republic Steel Corp. They are taconite development and foreign ores. Speaking at the annual meeting of the Minnesota Section of the AIME, Mr. Richards pointed out that with only ¾ of 1 pct of the land area of the world and with only 2 pct of the population, the states bordering the Great Lakes "do the stupendous job of producing over 35 pct of the steel made on the entire globe."

He emphasized the importance of the other industries existing in the area because of steel. While value of the products of the steel mills in these states is $6 billion per year, the value added by all manufacturing is more than eight times that figure or $50 billion per year, a considerable part of which is due directly and indirectly to the steel industry.

Protection of this total industry demands assurance of the present iron ore supply and increased production to keep pace with the growing demands for steel. Iron for these purposes must be supplemented both from beneficiation of the abundant taconite deposits and by constructing the St. Lawrence Seaway to permit import of foreign ore, he declared.

Mr. Richards noted that cost of facilities for taconite beneficiation was on the order of $30 to $40 per annual ton of pellets produced. For the projected 40 million ton per year capacity an investment of about $1.5 billion will be required. He added that taconite production will bring more fuel and electrical power to the region. Employment will rise with accompanying increases in personal income. Because the taconite industry will be a year-round operation a more uniform economy will result. "The people and their legislators should give the taconite industry confidence that its problems will be recognized and assurance that it will live in a sympathetic atmosphere."

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THE staccato beat of tom-toms cut through the Belgian Congo night. It was the primitive voice of Africa—relaying the news that a stranger had entered the Shinkolobwe area of the Belgian Congo. Quickly, smoothly, native militia mobilized. This is what happens when someone who doesn't belong wanders within range of the most fabulous uranium mine in the world.

Hundreds of tons of uranium ore come from the mine in the Katanga country. Security measures go to fantastic lengths to protect it as a source of pitchblende for the U. S., according to an article in the magazine *World*. A business man, inquiring about a road which happened to go through the Shinkolobwe Plateau, became the subject of a silent security check. The check started with a phone call by the desk clerk to whom the inquiry was directed. A thousand miles away, another kind of clerk checked through a file of suspected spies. The man was cleared, but not before an elaborate system of investigation had certified his status.

Until 1940, the mine was more or less a marginal proposition. Then huge orders for pitchblende began to come from the U. S. With Hiroshima, it became one of the most important places in the world—and every attempt was made to make the world forget it ever existed. Native workers never leave the housing zone except to go into the mine for the duration of their work contracts—3, 6, or 9 years. Work at the mine goes on round the clock. The amount of ore mined and shipped is kept secret. As a smoke screen nonfissionable materials are shipped with uranium ore.

Mechanized patrols are constantly on duty. Tall towers provide unobstructed fields of fire for crack marksmen. Barbed-wire fences which open only at cement guard houses surround the mine camp. Few outsiders have ever been permitted to inspect the mine and even a U. S. Ambassador was refused permission to see the operation.

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DESPITE the close relationship between American youth and the pseudo science of space travel, ray guns, and hot rods, results of a recent survey released by the National Education Assn., indicates that youngsters are passing up school science courses. They think them dull and unrealistic.

According to the survey taken in schools with a total enrollment of 326,000 students in 42 states, science teachers feel that the shortage of engineers could be alleviated if the importance of modern technology and the opportunities in that field were driven home more strongly to the students.

Why students don't choose science courses was a question posed by the survey. Some of the answers were: reluctance of students to take courses which are somewhat more difficult and which might lower their grades; lack of communication between high schools and colleges concerning science programs; little emphasis on science experience by teachers to students in grade schools. Following somewhat the same line, *Mineral Industries*, published by Pennsylvania State University, did a figurative dissection of mineral engineering students—seeking individual reasons for choice of career. Tom Falkie, a third semester student studying mining engineering, feels that for him, the choice was a natural one. Hard coal mining is the major industry of his home region. He believes that there are other opportunities in the mining field aside from the classic ones. Talking to mining people and visiting mines can lead to a research or sales career.

Bob Curran likes the active life that mining promises, and he wants to be a hard rock miner. George Trevorrow learned about mining first hand, working in mines during summers. George Schneider's father is a graduate engineer with the Pennsylvania State Dept. of Mines. Jim Reilly's father is in production work in central Pennsylvania.

Dean Porterfield, another student, chose geophysics because the course offered a heavy math and physics program. He's happy about his choice, and as graduation nears, feels that opportunities for jobs are good.

Dick Crosby, studying to be a mineral economist, chose his field because while he liked engineering, he did not want to go into the production end of mining. Most of the men who graduate as mineral economists start out in some other curriculum. They find somewhere along the way that their interests are less technical and more inclined to the financial end of the industry.

Mineral Industries states that "in the high schools of the state (Pennsylvania) students have little chance of studying in, or even hearing of, such exciting fields as geology, geochemistry, metallurgy, fuel technology, mineral economics, or mineral preparation."

Finally, from Harvard University, 23 prominent U. S. educators warn that the nation is critically near a breakdown in the first step of training scientists—high school science teaching. They find that the situation is "not only insupportable but perilous."

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BRINGING the second Errington underground mine and the new G orebody of Steep Rock Iron Mines into production will cost an additional \$5 million over current earnings and the \$8 million made available by Inland Steel Co. No more stock will be issued, it is reported, but the funds will be borrowed against production. Diamond drilling at the ends of the G orebody have revealed an additional 500 ft of ore. The orebody is at least 4500 ft long and an average of 160 ft wide.

On the 700-ft level of the Errington orebody from where initial production will come, two crosscuts have been driven across the orebody and into the footwall. A third crosscut is halfway across. One heading shows a continuous ore width of 223 ft, according to *The Northern Miner*. Water encountered on the 700-ft level has been negligible.

Conveyor belt haulage for the ore is functioning well. While it is premature to estimate costs, mining may be under \$2.50 per ton. Dredging and stripping at the Hogarth mine is making progress, with two dredges removing silt at a rate of 2 million cu yd per month. The job may be finished by late summer. More than 1 million tons of open pit ore will come from the Hogarth pit for the year's shipping period.

An all steel loading terminal is under construction at the Hogarth pit at a cost of \$750,000, with a railroad spur extending to the area. The terminal and the conveyor belt delivering ore from the crusher in the pit will not be ready until summer.

Orebody C, for which Inland Steel signed a lease, covers an area of about 1200 acres. While Inland's Caland Ore Co. has not actually started operations on this orebody, a \$25 million dredging contract has been given to Construction Aggregates Corp. The ore is under 150 ft of water. Even after the lake is drained, about 150 million cu yd of silt remain to be removed before the estimated 50 million tons of 53 pct Fe ore can be mined. The deposit, incidentally, dictates that underground methods be used.

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THE saying "It pays to be safe" is taken quite seriously by the employees of Hanna Coal Co., division of Pittsburgh Consolidation Coal Co. For them it happens to be literally true, as Hanna inaugurates another safety contest, with trophies, an automobile, a trip to Florida, and various other prizes. The contest is open to all United Mine Workers employees of Hanna. Trophies will be awarded to surface and underground mine operations with the lowest accident records. The winner of the automobile will be a man who has had no lost-time accidents during the contest period. He will drive off with a 1954 Chevrolet sedan fully equipped with radio, heater, and seat covers. Only one safe worker can win it.

The winner of the Florida trip will be the man who submits the best safety slogan—and his wife goes along too. There are a flock of individual awards for no lost-time accidents. Belts, cigarette lighters, electric clocks, wallets, and broilers have gone to winners in the past.

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GAW stands for guaranteed annual wage—and there are indications that GAW will be one of the main objectives of the CIO's United Steelworkers when the union begins contract negotiations in a few months. A recent issue of *Steel Labor*, the union's official publication, maintained that GAW is a "basic, practical way" to meet unemployment and to head off a "more serious downturn in the years to come."

The article continued that "just as the steelworkers pioneered the pension and social insurance programs that set the pattern for industry throughout the country, so we propose to lead the way in the fight against unemployment and depression with the guaranteed annual wage."

United Steelworkers made a demand for a guaranteed wage in 1952, but compromised by accepting a wage increase. A GAW proposal submitted to Aluminum Co. of America last year called for a weekly amount 30 times the standard hourly rate in the event of a worker's unemployment. It was turned down. The Steelworkers and the International Electric Workers, another CIO union, have joined in sending out questionnaires dealing with layoffs over the past 16 years to leading corporations. Among those receiving the questionnaires are U. S. Steel Corp., Bethlehem Steel Corp., and General Electric Co.

Drift of Things

REVOLUTIONS are not always political, sometimes the industrial ones have the greatest impact on living standards and productivity. One small but important revolution is now underway in drilling. Without a great deal of fanfare, unless studious cost and performance reports are fanfare, the whole mining industry has taken to harder hitting, lighter, more mobile drilling equipment both underground and in the open pit. With wider use of airleg drills it may be said that the *Mexican Jumbo* has been polished up and powered. And all these changes mean more efficiency, of course.

Every item in the complex of tools that produces a drill hole has undergone the closest scrutiny, from the cutting edge of the bit back to the air hose. In this issue *ME* features two articles, one on drill steel, and the other on tipping steel with tungsten carbide bits. Coming up soon are other drilling articles, on rotary drilling, on percussion drilling, and an especially interesting one on Swedish drilling practice. The latter article goes one step beyond drilling, turning attention to improved loading technique for the answer to the problem of getting enough powder into the face when using smaller, easier-to-drill holes.

Considering the range of articles now scheduled for publication brings to mind the job done by the committees that obtain, study, and review these papers. In every organization there are small, hard working groups that perform tasks whose importance is in inverse proportion to the credit they receive. One such group within the AIME is the Divisional Publications Committee, of which more below. —R.A.B.

THE Divisional Publications Committees perform one of the most important functions of the Institute. They have the very high purpose of examining the papers submitted to the AIME and approving them for publication, subject to the final approval of the Technical Publications Committee.

The Committees act as readers or select readers who are experts in the field covered by the paper. These members are not editors, but are the critics of the scientific information included in the paper. When this panel of experts has completed its review, the manuscripts selected should be the latest and most correct original information available on the subject.

The Institute, and therefore the whole of the membership, is dependent upon the Divisional Publication Committees to see that material published by AIME is of unquestioned technical accuracy, and that high professional standards are maintained, regardless of the position of the author. These expert readers must be alert in protecting the membership from the banal article, the self-advertisement, and from the rehashed review, as well as active in finding and pushing forward the vital, original technical information that the professional mining man must have to help his industry grow stronger. This is the Committees' responsibility.—C.M.C.

WITH mining men coming into New York for the Annual Meeting (as this is written) we would like to suggest that one place the underground miner might find familiar atmosphere is the Public Library. Now this is not as farfetched as might appear. It seems that the ubiquitous miner's cap lamp has been temporarily adopted as garb for the well-dressed book courier, that is while the stacks are blacked out during installation of modern fluorescent lighting. Then too, Macy's Department Store must have heard about the Annual Meeting, for it recently pulled the main display out of the famed 34th Street windows. Several mining people had been known to look longingly at this display of about $70,000 worth of silver bullion in bars. A lot of the public had been unimpressed by the slightly dirty looking ingots, but with metallurgists coming to New York in force . . . well.

IN a little while residents around Rockdale, Texas will be talking about the big one that got away—in a spot that a few years ago was, to say the least an unlikely place for fish—no water. But by June 1, the lake that Aluminum Co. of America built to supply water to cool generators sending power to the four potlines at the firm's 170 million lb per year plant, will be opened and well stocked. About 18,000 black bass and 5000 finger-length bream will be placed in the lake by the Texas Game and Fish Commission.

Some 1.38 million cu yd of earth were removed to build the mile-long dam which forms part of the lake's shoreline. A dike separates intake and outlet parts of the lake with a canal connecting the two sections. Temperature of lake water will never reach high enough to harm the fish and by the time cooling water is returned it will be purer than the original streams. Average lake depth is 18.29 ft. (M. A. M.)

A RHODE ISLAND motorist sheepishly swears this story is true, we make no guarantee of accuracy, but nominate it as the news story of the year.

This driver was on the Merritt Parkway going toward New York when his car stalled because of a dead battery.

He flagged down a woman driver, and she offered to give him a push to start his car. Because his car has an automatic transmission he explained that "you'll have to get up to 30 to 35 mph in order to get me started."

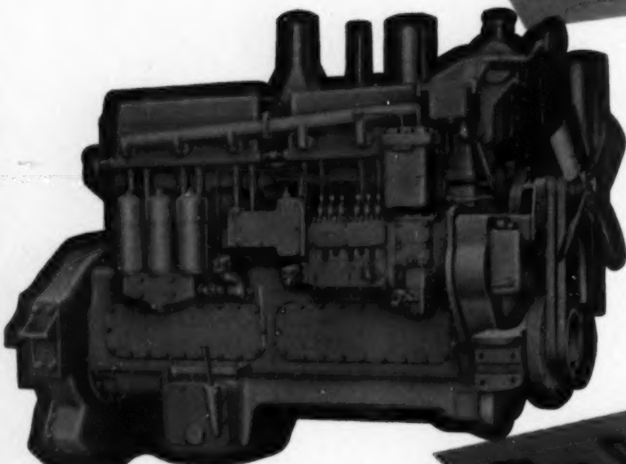
The lady nodded wisely. The stalled driver climbed into his car and waited for her to line up behind him. He waited and waited. Then he turned around to see where she was.

She was right there all right—coming at him at 30 to 35 mph.

The crash caused $300 damage to his car.

The moral: A mine is one of the few places that is still a man's world.

The Powerful D8 Engine



The new high performance of the Caterpillar D8 Tractor stems from the 185 HP, 1200 RPM diesel engine. Important new features, built into the time-tested and job-proved Caterpillar design, contribute to longer engine life and decreased operating costs as well as higher power. See your Caterpillar Dealer for all the reasons why the D8 is a better value.

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Storke Level Operation Makes Climax N. America's Biggest Underground Mine

by E. J. Eisenach and Edward Matsen

AT the present time the Climax Molybdenum Co. is the largest molybdenum producer in the world and the operator of the largest underground mine in North America. It has grown steadily and rapidly since its beginning in 1917. Only 250,000 lb of MoS_2 were produced during the first year from 1917 to 1918 when the company was formed. Operations were suspended shortly thereafter because of the lack of a market and knowledge of the uses of molybdenum. During World War I it was used as a substitute for some rare metals and for minor chemical purposes. After 5 years of intensive research and sales promotion, Climax metallurgists succeeded in finding proper applications for the newly discovered metal and the mine was reopened in 1924. Demand for molybdenum increased gradually and in 1930, 3.5 million lb of MoS_2 were produced. In the meantime, research work for additional uses of molybdenum continued until it had established itself as one of the most important alloying elements in producing high quality steels. After the start of World War II, from January 1942 to October 1953, 43 million tons of molybdenum ore were mined and 484 million lb of MoS_2 were produced.

Climax mine is located in Colorado, 100 miles southwest of Denver and 12 miles northeast of Leadville on the Continental Divide of the Rocky Mountains. The principal mine workings range from 11,170 to 12,000 ft in elevation. Topographically, it lies in the base of a large cirque that is surrounded by Mt. Bartlett to the north, Clinton Peak and McNamee Peak to the east and Ceresco Ridge to the south. This cirque opens to the west and to the Ten Mile Valley.

Geology

The Climax orebody lies in the mineral belt that traverses western Colorado from the San Juan Mountains northeast to the Boulder County district. With its essentially molybdenite mineralization, Climax is mineralogically unique compared to the surrounding mining districts of Leadville, Kokomo, Alma, and Breckenridge, all of which are characterized by base metal deposits carrying silver, gold, lead, and zinc.

The orebody occurs in the western portion of a narrow north-trending fault block of pre-Cambrian rocks bounded on the west by the north-trending Mosquito Fault. This fault has a normal displacement, west side down, of some 3000 to 5000 ft. Movement along the fault has brought pre-Cambrian rocks on the east into juxtaposition with Paleozoic sedimentary rocks on the west. Quartz, biotite, schist, gneiss, and granite comprise the pre-Cambrian terrain, and are cut by dikes of tertiary diorite, quartz, monzonite and rhyolite porphyry.

E. J. EISENACH and E. MATSEN are respectively Asst. Mine Superintendent, and General Mine Foreman(Storke Level), Climax Molybdenum Co., Climax, Colo.

This is the Storke Level portal during the final phases of development.

Ideally, the main orebody may be likened to an inverted half cantaloupe slightly elongated in a northeast direction. It domes over to form an outside apex approximately 1100 ft above the Storke Level. Underneath the east portion lies another similar, but smaller ore zone called the inner ore body. The two ore zones have a common east margin. Only the northern and western portion of the main orebody persists down to the Storke Level. Here the main ore zone is slightly arcuate in shape, concave to the east, about 2200 ft long from south to north, and varies in width from 300 ft on the north to 500 ft in the center, and tapers to a point on the south. Ore boundaries are based on assays of diamond drill core and drift samples. Overall, the boundaries are rather uniform although local irregularities are common. Some adjustment of mining layout has been necessary to compensate for displacement of the ore zone along the west shear zone and the location of the Storke Level itself was influenced by the irregularities in the main ore zone as it pinched out along the south side.

Molybdenite is the predominant ore mineral and occurs as a fine grained fracture filling with quartz in veinlets that cut the host rock randomly. Pyrite, topaz, wolframite, and chalcopyrite are also found in random veinlets associated with the molybdenite. A small amount of monazite and cassiterite occur disseminated through the rock. Some fluorite and lead-zinc mineralization of definitely post-molybdenite age is found throughout the orebody. Oxidation of the molybdenite has been important only within that part of the orebody from surface to 100 ft below surface, although a small amount of molybdite, the oxide, is found near the Storke Level.

Since the molybdenite is a fracture-filling, the grade of ore is largely dependent upon the amount of pre-ore fracturing. Throughout the ore zone all

pre-ore rocks are intensely fractured. It is difficult to find a cube of rock 2 in. on edge that does not contain at least one fracture. As a result, many local areas of unstable ground were encountered. Gouge filled post-ore fractures associated with the west shear zone and the Mosquito Fault are especially conducive to unfavorable ground conditions.

Rocks encountered on the Storke Level are pre-Cambrian schist, gneiss and granite, dikes of tertiary quartz-monzonite and rhyolite-porphyry, and a mass of replacement rock called Climax porphyry. With the exception of the post-molybdenite-rhyolite-porphyry, all rocks are quite altered. In the ore zone the rocks are intensely argillized and sericitized. Silica veinlets are abundant throughout but increase toward the east side, or footwall, of the ore body where the rock contains as much as 90 pct silica.

Cavability of Ore

Observation of various rock fractures and characteristics in different occurrences of cementation and separation, also early mining practices by shrinkage stoping and caving pillars on Leal and White Level led to the conclusion that some form of caving operations would be applicable for the mining of Climax ore. As a result, a panel caving system was introduced on the Phillipson Level using the chute and grizzly system. Although the chute and grizzly method increased the production considerably, it also brought out deficiencies and difficulties in safety and efficiency of handling caved ore.

A study of handling ore with slushers was made on the White Level on a small scale in 1934 and 1935. The results of this experimentation convinced the company to further these tests with slushers on a larger scale on the Phillipson Level for comparison with the chute and grizzly system. Records on the Phillipson Level showed the handling of twice the amount of ore with the slushers and at the same time lost time accidents were reduced to one third as compared with chutes and grizzlies. In general terms, the main reasons for eliminating the chute and grizzly methods and converting to slushers were: (1) Decreased overall costs; (2) Increased production; (3) Improved efficiency; (4) Better ventilation, safety and communications; (5) Easier maintenance and service to stopes and working places: (6) Better draw control.

The management was convinced of the benefit of this change and has carried on an extensive and continuous testing program to further improve the slusher system. The Storke Level was planned in its entirety for the use of the slusher system of production with all the knowledge and modification available at the time.

History of Storke Level

As early as 1938 the management realized the necessity of developing a lower level for future production. Previous exploration from the Phillipson Level indicated that the ore extended downward over 500 ft with a good grade and a large area. In the early 1930's, a 500-ft shaft was sunk from the Phillipson Level on the westerly side of the orebody and 500 ft of drift was driven at the bottom of the shaft to obtain water for the mill. Diamond drill holes were added horizontally and downward to increase this water supply. Assays of core from these holes verified the large ore reserves below the Phillipson Level. As a result, further exploration of this level was started. From September 1937 to March 1940, 14,000 ft of 5x7-ft drifting was completed and an inclined raise was connected with the Phillipson Level on the south footwall. This raise had a twofold purpose, ventilation and recirculation of warm air from the lower level to Phillipson Level, and as a second entrance to the 500 level.

In the meantime, the demand for molybdenum was increasing and exploration work was speeded up to outline the ore reserves below the Phillipson Level. An extensive diamond drill program was started and an intermediate level was driven 250 ft below the Phillipson to locate the ore boundaries. Available information was compiled and study was made to decide and establish the new mining level.

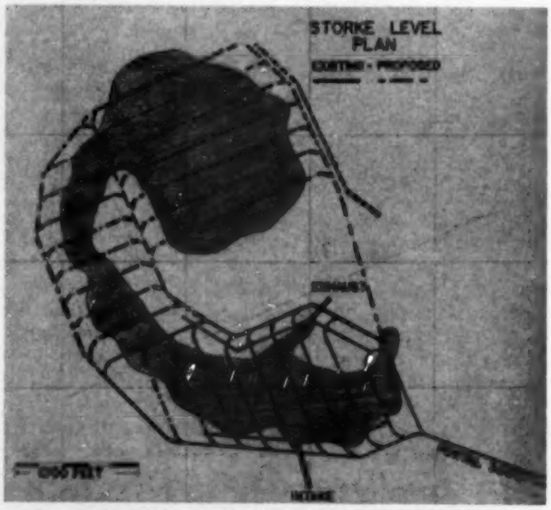
The items to be considered were: (1) Cave control; (2) Shape of orebody as various elevations; (3) Control and recovery of ore; (4) Water drainage; (5) Flexibility of development and production; (6) Economics.

It was decided that the 300-ft level below the Phillipson would give the best mining conditions and be the most adaptable for development.

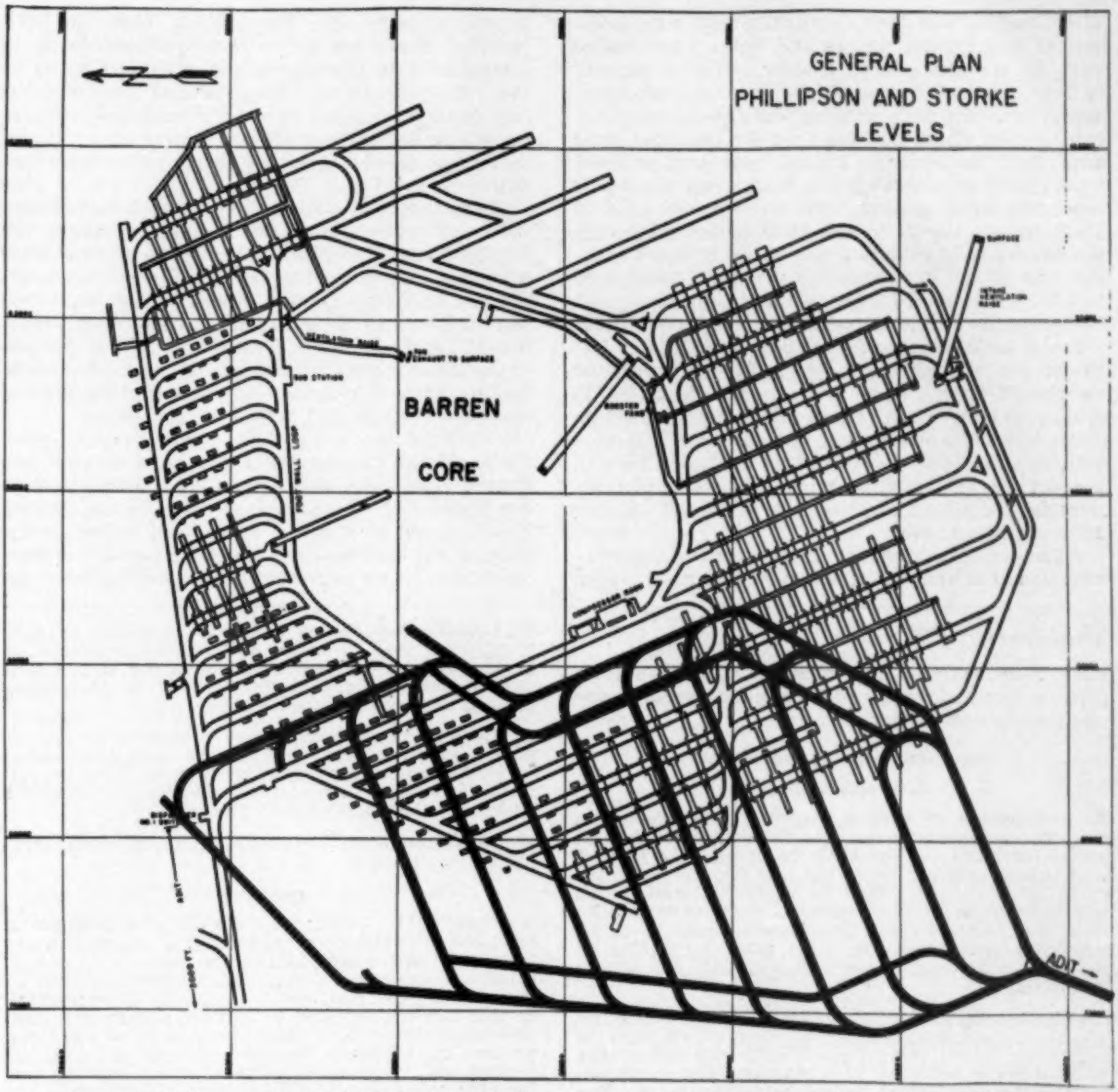
After intensive study of the costs, access, and a potentially rapid increase in development, the decision was made in 1948 to gain entrance to the ore zone by the means of a Storke Level adit. The existing ore crushing facilities were not adequate for future increased tonnage. It was decided in 1950 to construct a new outside crusher and convey the crushed ore over the surface to the mill. (For a discussion of the alternatives, see MINING ENGINEERING, Jan. 1953, p 36.)

With the developed ore reserves greatly depleted under the war time demand in years 1942 to 1946, followed by the unstable world political situation, plans had to be made to develop new ore reserves which could be available on short notice. Development work had dropped abruptly during these years and the company had less reserve broken tonnage in 1948 than at any time during the preceding 10 years.

The Storke Level tunnel was started in the fall of 1948 by an outside contractor. This work included 2900 ft of 15x13-ft tunnel, 6000 ft of hangingwall, footwall and haulage drifts to the orebody and 2500 ft of main ventilation drifts. This work was completed in 1949 and no additional development was done on this level until 1951.



Plan shows the possible extension of the Storke Level workings to tap further reserves.



Storke Level, shown solid black, is 300 ft below Phillipson. Storke adit surfaces on eastern slope of Continental Divide.

Expansion of production and development work was started due to the critical world situation in the spring of 1951. Lack of available manpower and shortage of housing facilities induced the Climax Molybdenum Co. to contract this development work with an outside construction firm. Development plans and specifications were prepared by company engineers. These plans were then submitted for bids and a contract was awarded on a unit price basis for this work. A company project engineer was assigned to inspect and oversee this work as to schedule, performance, and safety of operations. He was assisted by inspectors and engineering survey crews totaling 15 men at the peak of the work.

The new contract was a continuation of the work done under the previous (1948) contract and covered the development of the southwest ore zone on the Storke Level. This operation consisted of: (1) Excavation of main loading drifts with timber or steel supports; (2) Excavation of hanging wall and footwall drifts; (3) Excavation of small ventilation drifts; (4) Excavation of slusher drifts and ventilation connections; (5) Excavation of intake and exhaust ventilation raises; (6) Excavation of intake and exhaust overcasts at intersections of loading drifts and main vent drifts; (7) Concreting of slusher drifts, overcasts and other miscellaneous concrete construction; (8) Roof bolting and *shotcreting* in weak and unstable rock excavations.

This contract work was nearly completed in 1951 and supplementary contract for additional work was agreed upon in January 1952. This consisted of additional haulage and loading drifts, large and small ventilation drifts, slusher drifts, ventilation connections, overcasts, and concreting. Undercutting, boundary cutoff, longholing, and blasting operations were also included in the supplementary contract.

The fringe footwall and hanging wall haulage drifts running in a northerly direction were excavated for supported sections 15 ft wide by 13 ft high, and for unsupported sections 13x11 ft. Parallel loading drifts of the same size were driven 200 ft apart in an east-west direction through the orebody.

The drilling setup in these drifts consisted of a 4-machine jumbo with 3½ in. Leyners on 4-ft shells, using 1⅛ in. round steel with 1¾ in. carbide

bits. Blasting was done electrically with both millisecond and regular delays and holes were loaded with 45 pct semigelatin powder. Timber support had to be carried close to the face and occasionally crown-bars with cribbing were necessary. Oregon lumber, 12x12 in., was used for caps and posts with 3x12-in. lagging. Bridge caps and cribbing were placed over these sets in some cases to support especially weak ground. Sets were placed on 5 to 10-ft centers except in the loading drifts between the slusher drift cutouts where predetermined spacing was used. Prefabricated structural steel arch sets were also used for haulageway ground support but were discontinued because of local difficulties.

Small ventilation drifts were driven midway between the loading drifts from the main exhaust ventilation drift. They were mined to 35-sq ft opening, at the same rail elevation as the haulage drifts and under the ends of the slusher drifts. Sets with 8x8-in. timber were placed in these drifts to support the ground that might become loose from secondary blasting and concussion in the slusher drifts directly above.

After completion of the loading drifts, the construction of slusher drifts was started at right angles to and directly over the loading drifts on 34-ft centers. These are single ended slusher drifts as compared with the double ended slusher drifts on the Phillipson Level. Single ended slusher drifts improved the slusher operator's visibility, ventilation, draw control, and slusher maintenance. It also permitted the handling of large rocks from the coarse cave.

The sequence of mining for the slusher drifts began with the excavation of the loading cutouts to the required size. This space is needed to accommodate a loading hopper, ventilation lean-to, manway landing and crawl-beam. The cutout is then concreted with columns to the sill of the loading drift with a manway and carspotter platform built into the side of the loading drift. Prefabricated steel sidegirders, loading aprons, a drawhole frame and supporting members are installed before the concreting.

Following the completion of the cutout, excavation of the slusher drifts and hoist cutouts was started. The drifts were driven 112 ft in length from the center line of the haulage drifts in the loading direction and 26 ft in the opposite direction for the hoist cutout that required 10x14-ft openings. Sixty-hp slusher hoists were set up for handling develop-

Shotcrete

Here is an abstract of the technical requirements Climax established for placing pneumatically applied mortar under contract.

Pneumatically Applied Mortar

Construction Methods

A Preparation of Surface. Before placing shotcrete, all loose rocks shall be removed from the surface to be coated, and the surface shall be thoroughly washed with a high pressure jet of not less than 90 psi to remove all clay, rock, dust or other material which would interfere with bonding the shotcrete to the rock. To facilitate bond, the cleaned surface shall be moistened not more than 1 hr prior to placing of shotcrete and excess water removed before shotcrete is applied.

B Placing. Shotcrete shall be placed under pneumatic pressure under the direction of men experienced in such work, with a cement gun of approved design, and in accordance with the best practice. The minimum thickness applied shall not be less than ½ in. The pressure in the placing machine shall be maintained at not less than 35 psi for hose lengths up to 100 ft, and shall be increased by at least 5 psi for each additional 50 ft of hose or fraction thereof. The water shall be maintained at a uniform pressure of at least 15 psi greater than the pressure in the placing machine. The Contractor shall furnish, install and maintain in satisfactory operation suitable gages for indicating air and water pressures.
In the shooting of all surfaces, the nozzle of the cement gun shall be held at such distance and position that the stream of flowing material shall impinge as nearly as possible at right angles to the surface being covered. Layers of shotcrete shall be applied at locations in the drifts, overcasts and raises as indicated on the drawings or as ordered by the engineer, but regardless of any use of shotcrete to obviate the necessity for temporary or permanent timbering to protect against spalling or slabbing off of rock, such orders or approvals shall not relieve the Contractor from his responsibility to maintain the safety of the support of the drift, overcast or raise until after the work has been completed and accepted by the Company.
Where shotcrete is to be applied in two layers, the time interval between successive applications of shotcrete must be sufficient to allow initial but not final set to develop. At the time the initial set is developing, the surface shall be lightly and carefully broomed to remove any laitance and provide a better bond with succeeding applications.

F Construction Joints. Construction joints or days work joints shall be sloped off to a thin, regular edge. Before placing the adjoining work, the sloped portion in the adjacent shotcrete shall be thoroughly cleaned as necessary, then wetted.

G Curing Pneumatically Applied Mortar. As soon as pneumatically applied mortar has hardened sufficiently, it shall be cured by application of Hunts Process membrane curing compound as elsewhere provided for under the General Concrete Specifications.

H No shotcrete shall be placed across openings carrying a flow of water.

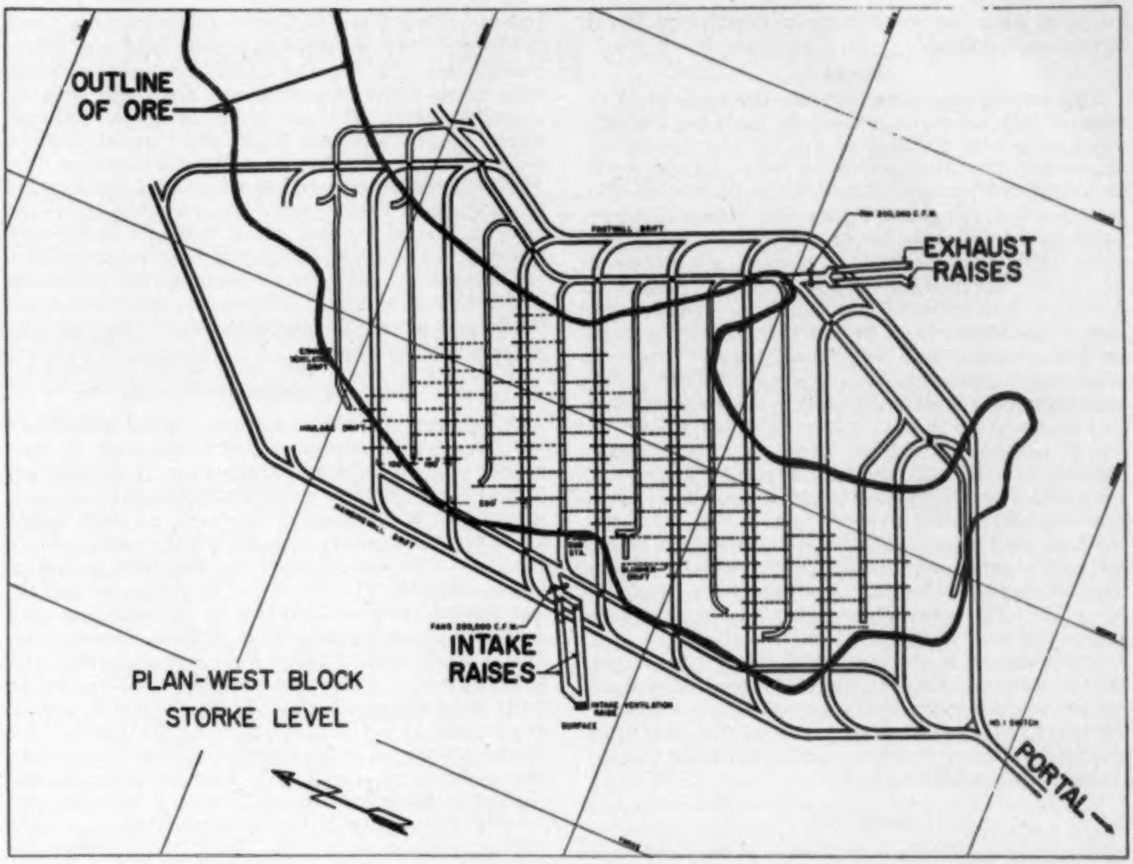
Materials

A General. Shotcrete shall consist of a mixture of sand and Portland Cement thoroughly mixed in a dry state, with subsequent addition of water. One part of cement to three and one half (3½) parts of sand by volume shall be used, and the quantity of water shall be that amount required to produce a mortar of suitable consistency and uniformity with a minimum amount of rebound. Machine mixing of sand and cement will be required. The minimum mixing time for each batch will be not less than 1½ min after the sand and cement are in the drum, and each batch shall be entirely discharged before recharging is begun. The mixture of sand and cement will not be used if it has remained in the placing machine or hopper over 3 hr.

D Sand for Pneumatically Applied Mortar. The term "sand" is used to designate aggregate in which the maximum size of particles is three-sixteenths of an inch. The sand shall consist of hard, dense, durable, uncoated rock fragments and shall in general conform to the requirements for Fine Aggregate as set forth in the General Concrete Specifications.
The sand used for pneumatically applied mortar shall not contain less than 3 pct nor more than 5 pct of moisture at the time of mixing with cement.

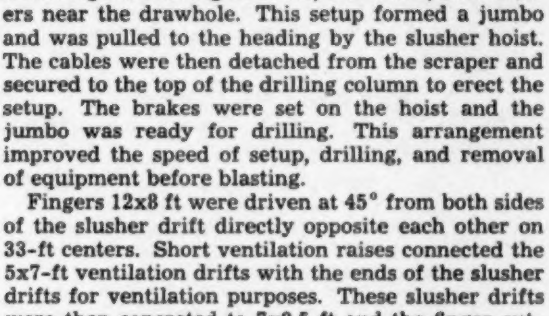
E Temperatures for Pneumatically Applied Mortar. Temperatures for placing of pneumatically applied mortar shall be the same as those set forth in the concrete section of these specifications.

F Admixtures. Should the contractor desire to use an admixture in the shotcrete for this work, he shall submit to the engineer full information regarding the product which is proposed for use.
(Sand used was screened —1/16 in., and moisture reduced below 5 pct by heated rotary mixers. Cement was added following drying operation, mixed, and transported to the working place. Here it was rescreened before being added to the type 2 cement gunite machine.)



ment muck. Scrapers were equipped with single horizontal bars between the arms for the purpose of attaching the drilling columns, crossarms, and drifters near the drawhole. This setup formed a jumbo and was pulled to the heading by the slusher hoist. The cables were then detached from the scraper and secured to the top of the drilling column to erect the setup. The brakes were set on the hoist and the jumbo was ready for drilling. This arrangement improved the speed of setup, drilling, and removal of equipment before blasting.

Fingers 12x8 ft were driven at 45° from both sides of the slusher drift directly opposite each other on 33-ft centers. Short ventilation raises connected the 5x7-ft ventilation drifts with the ends of the slusher drifts for ventilation purposes. These slusher drifts were then concreted to 7x9.5 ft and the finger cut-



Longitudinal section shows concreted slusher cutout employed on Storke Level.

outs were concreted to 8x4.5 ft. The thickness of concrete lining in the slusher drifts varied from 1 to 4 ft depending on the points of most wear. The completed slusher drift averaged about 800 cu yd of concrete. Reinforcing steel is restricted to the loading and hoist cutout stations.

Two parallel intake ventilation raises were driven on a 55° incline from the main intake ventilation drift to surface with a 20-ft pillar between them. These raises were timbered by square sets and cribbing. The combined area of these raises is 152 sq ft.

Two exhaust ventilation raises were driven in a similar manner from the exhaust ventilation drift to the Phillipson Level. The combined area of these two raises is 230 sq ft. The main airway raise was roof-bolted and shotcreted rather than timbered to eliminate resistance.

Fifteen intake and exhaust ventilation overcasts were constructed over main loading drifts varying in size from 35 sq ft to 225 sq ft of area. The arches and ribs were roofbolted and shotcreted in all overcasts. The structures of the overcasts were built of reinforced concrete in such a manner that the flow of air can be regulated.

Weak and otherwise unsupported rock on the Storke Level was roof-bolted and shotcreted rather extensively. The Storke Level adit was shotcreted for nearly its entire length. The main underground powder magazine and transformer station were shotcreted. Some roof bolting was necessary to secure the arch of the loading drift in slusher cutouts and overcast excavations and also to hold brows in slusher drifts and fingers before concreting. Approximately 5000 wedge type roof bolts and 60,000

cu ft of shotcrete were used in the Storke Level development work.

Stoping

The stoping operation includes the excavation of finger raises, sidelines, undercuts, dogholes, and the longholing and blasting of the pillars formed by this work. The total excavation before longholing is 55 pct of the horizontal area to be undercut. The sequence of mining is to extend the concreted finger raises up on 45° slope to the drawpoint beneath the cave. From here sidelines, 8 ft wide, are driven at 90° to the finger raises on a 45° incline to meet at an apex with adjacent sidelines. At the sideline apexes undercuts 10 ft in width are mined normal to the sidelines both ways on plus 45° slope to apexes with undercuts from opposite directions. The apexes over the slusher drifts form the major apexes, and between the slusher drifts the minor apexes. A 6x6-ft opening is driven along the major apex, directly over the slusher drift to reduce the size of the pillar and allow more area for the swell while blasting stope pillars.

Along the hanging and footwall at the edge of the orebody, a cutoff shrink line is established by driving 7x9 ft raises at regular intervals to the height of 50 or 75 ft above the stope and pillars thus formed are drilled for blasting.

When stoping is discontinued inside the boundary of the orebody, a boundary cutoff is driven from the top of the undercut through the pillar parallel to the line of the slusher drift to reduce the possibility of drilling into missed holes when resuming undercutting in the adjoining stopes.

Longholing

The drilling pattern of longholing was arranged to afford the maximum breakage of the pillars. Four holes were drilled in vertical arrangement and 3-ft horizontal spacing was used along the slope of the undercut and parallel to the line of the slusher drift. The top hole acts as a *buster* of the back while the remaining three holes are used to remove the pillar. In well-fractured ground the top holes can be eliminated. Extra holes were drilled over the sidelines and draw-point to help start the caving action.

Blasting

The loading of the pillars in the Storke Level stopes was supervised very closely. A retreat system from the hangingwall cutoff toward the footwall was used, and a rigid inspection was made after each row of pillars was blasted. Portions of the pillars or stubs that may have been left had to be removed before the next blast. Millisecond electric caps were used entirely for all 72 stopes caved. The holes were connected in parallel series; each series contained from 12 to 20 holes. Size No. 8 to 12 leadwire was used for firing from the 440-v ac circuit.

Ventilation

Air is brought down from the surface to the Storke Level by means of a 96-in. Axivane type fan delivering 200,000 cfm. With variable pitch blades the amount of the air can be regulated to the desired volume. The air is discharged into the main intake ventilation drift on the hangingwall side and distributed to the loading drifts by means of controlling openings at the overcasts. The fresh air flows through the loading drifts into the slusher drifts through the lean-to and the drawhole.

At the opposite end of the ventilation system a 108-in. Axivane fan pulls the contaminated air out of slusher drifts by way of slusher drift ventilation connections and via the small and large exhaust ventilation drifts and overcasts, from the hanging to the footwall. This air is forced up the exhaust raises to the Phillipson Level and through the old workings on the Phillipson Level to the surface. The ventilation system is regulated so that the amount of air at intake end is greater than at discharge, thus causing the air to flow out of the adit to prevent freezing. Proper ventilation has been recognized by the company as one of the most important phases in production and safety. Continuous experiments in the improvement of ventilation and industrial hygiene are being made.

Production

In February 1953 the company started withdrawing ore from the caved stopes in the area of contractor's work. As the contractor's work progressed, and new stopes were made available, the company mechanical and timbering crews moved in to install the slusher equipment and prepare the slusher drifts for draw. To control the new cave area properly, is was necessary to move slusher hoists frequently and special consideration had to be made for easy assembling and disassembling of slusher equipment. The 2-drum tandem hoists were especially designed to be adaptable to Climax operations. The slushing units were equipped with 150-hp high-slip motors developing 17,000-lb rope pull at 280 fpm. The production schedule for a period of from 2 to 3 years was planned for 10,000 tpd. This has been accomplished by operating two shifts with a 20-man crew on each shift. Four trains hauling 160 tons per trip are operated by each crew. Slight track grades are provided for haulage with all loaded trains moving downgrade. The movement and distribution of trains are controlled by electric signals and carrier telephone dispatching system. Twenty-three 150-hp slusher hoists are in use on the Storke Level and the production crew rate is averaging 240 tons of ore per manshift.

Work Done Under Contract

The tremendous amount of work accomplished during the contract consisted of:

1. 2000 ft of main haulage drifts
2. 7500 ft of main haulage drifts
3. 700 ft of large ventilation drifts
4. 6000 ft of small ventilation drifts
5. 1800 ft of ventilation raises
6. 11,000 ft of slusher drifts
7. 500,000 sq ft of undercutting
8. 74,000 cu yd of concrete placed

Careful planning and engineering, correlated from previous experiences at Climax, were the most essential factors which contributed to the success of this enterprise in a relatively short time. Under the strict specifications and limited but efficient company supervision, this work was brought to completion without any interruption in the work schedule or disagreement between the contracting parties. Completion of this large underground development program will permit the Climax Molybdenum Co. to meet any increased production requirements in the event of a national emergency.

The authors wish to express appreciation to the management and the engineering staff of the Climax Molybdenum Co. for their assistance and support in preparing this paper.

Pressure Grouting At Deep Creek

by A. V. Quine

I wonder if anyone present has ever watched 150 ft of diamond drill rods come crashing back through the machine, snapping off like matchsticks against the side of the drift? Or, perhaps has stood on a raise staging, 25 ft above the level as a buzzy steel suddenly tapped a 250 psi sleeping giant—making necessary three months work before the drill steel could be released from the chuck?

*Water, in its gentler moods, is a fit subject for romance and poetry, but I can assure you that far more lurid words were used to describe this ripping, tearing, roaring monster. The near presence of such terrific forces, capable of producing sufficient volumes of water as to strain the capacity of the pumps, has not been conducive to untroubled dreams at the Deep Creek mine.**

* Quoted from author's oral presentation at Northwest Mining Convention.

FOUR years ago several mining operators shook their heads and predicted that Goldfield Consolidated Mines Co. would never be able to mine the zinc-lead ores from the 550 to 650 level areas of the Deep Creek mine. Their prediction was not without apparent justification, for everywhere one gazed, water was spurting or dripping; every hole that was drilled produced more and more water under pressures of 220 to 250 psi, all from apparently unrelated channels. When the volume of water from one level alone reached 2200 gpm with a possible flow of 5000 to 6000 gpm if the area were fully tapped, development work was brought to a halt and the few holes into the water area were grouted off under contract. As zinc prices had completely deteriorated, the workings were allowed to flood.

Upon reopening the mine, in late 1950, it was discovered that the relatively few sacks of cement used in the previous grouting operation had done a tremendous amount of good. Consequently it was decided to systematically drill test holes with a longhole machine and diamond drill and to seal all holes with a cement grout solution. After 30,000 ft of drilling, and the pumping of over 20,000 sacks of cement, 15,000 of which were placed in the supposedly impossible 650 level, the company was successful in literally forcing the water out of the ore zones and opening up large tonnages for subsequent extraction.

Structurally, the ore deposits lie within steeply dripping, parallel shear zones in dolomite marble, overlain by as much as 350 ft of glacial valley debris, the latter no doubt forming a vast reservoir of water below the hardpan bed of the present stream flowing through Deep Creek Valley. At irregular intervals steeply dipping faults appear in the ore zones, sometimes parallel and sometimes transverse to the trend of the orebodies. As the ancient glaciers have truncated the tops of the fault zones, the latter form ideal channelways for the percolation of surface waters. The word *percolation* is used advisedly, for wherever these channelways have been tapped one cannot in truth say that the water percolated into the heading.

These channelways are rather remarkable in themselves. Where contacted at the Deep Creek mine some of the passageways are actually washed out portions of the fault fissure, as much as 6 to 8 in. wide, and usually, when first contacted, spew forth considerable quantities of fine sand and crushed rock. However, many of the channelways seem to be of the *conduit* type, which appear to be pipe-like solution channels carved out of the solid rock immediately adjacent to the fault zone, and also lying within the more solid portions of the larger fault areas. These conduits are apparently completely unrelated, as in the more than 300 holes drilled, tapping many channels, only one hole showed any appreciable return from any other hole, even though at times some holes were within a foot of others. What is still more remarkable is that no return of cement or coloring material was found in any other spot (except one) in the mine. Where the cement grout or colored test water went is a complete mystery, even now; what is even more mysterious is the actual formation of the conduits which presupposes considerable velocity of movement as well as the presence of the known hydrostatic pressure. At 650 ft vertical depth, where could the water be going? It is reasonable to presume ancient channelways cut along the bottom of the present glacial valley, which eventually find an outlet in the deep gravels of the Columbia River, 7 miles distant.

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There is nothing particularly difficult in the process of grouting. Perhaps its very simplicity has been a prime factor in the lack of published information concerning its application in the mining field. Every application differs in some respect, structurally, feasibility or economically. However, the fundamental principles still remain the same: a mixture of cement and water must be pumped with more or less pressure into fractures, faults, conduits or other broken and brecciated areas for the purpose of either halting water flow or of adding strength to the surrounding rock, or both.

Deep Creek's Grouting Technique

Probable water areas are tapped by two conventional methods: diamond drill and longholing with a 4-in. drifter.

Where relatively high pressures (250 psi) are expected, a badly fractured zone is to be crossed, or lengthy upholes are to be drilled, the diamond drill is used to advantage. An EX hole is collared and drilled for 5 ft after which the hole is reamed to slightly more than 1½-in. diam. A 3-ft length of 1½-in. pipe, threaded one end, is inserted in the hole, the latter being filled with a hand-mixed grout solution composed of one part Lumnite quick-set cement, and three parts Portland cement. After solidifying, which takes place within ½ hr, a 2-in.

gate valve is fitted to the 1½-in. pipe protruding from the hole, the EX bit is inserted through the valve and drilling is continued until the first water zone is reached. The rods are removed, the valve closed, the grouter hooked on, and the hole plugged with cement. After a 3 to 4 day setup period the hole is redrilled and the same procedure followed until the wet zone has been completely sealed off. Naturally, in actual practice a series of holes are drilled and capped with valves in order that the diamond drill and the grouter can work to better efficiency. If pressures greater than 250 psi are expected, it would probably be necessary to fit a packing gland on the drill setup, although it has never been required to do so at Deep Creek.

The great majority of the test holes have been drilled with a 4-in. drifter, using at present 1-in. hexagon alloy steel in 3-in. coupled sections with separate water swivel and 2-in. Carset bits. After water has been struck and the rods removed from the hole, a 3 to 5-ft piece of 1½-in. pipe (length dependent on the fractured condition of the rock) is placed in the hole and secured by means of wooden wedges. Many operators prefer to use a two-piece apparatus fitted with an expandable rubber washer called a packer, but experience at Deep Creek has caused reliance almost entirely on the wedged pipe for the larger, high volume holes and use of a packer for water contacted with either the 1⅝-in. stopers, the 1⅜-in. jackleg drifters or for those holes running only a small amount of water. Under high pressures the packer has been found to be more difficult to install than the conventional 1½-in. pipe, and, if any distinction can be drawn between being wet and being wetter, not quite so much water is sprayed upon the installing crew by the latter method. After installation of the pipe, a 2-in. gate valve is easily fitted and the valve closed ready for grouting.

Present drifting and crosscut practice in unknown or probable wet areas require the drilling of two, 10-ft advance holes, one on each side of the heading, as a precaution during the drilling of each round. If water of any consequence is encountered, the heading is stopped until further long hole drilling and subsequent grouting has been completed.

The Pumps

The actual pumping of the grout solution is accomplished by the use of a Gardner-Denver 6x3x6 Duplex high pressure steam pump, fitted with special rubber valve inserts, special alloy liners reducing the diameter of the liquid cylinders to 2½ in. and special alloy piston rods fitted with 2½-in. rubber pistons. A stand-by 7x3x10 pump fitted out the same way is held in reserve. With this equipment pumping pressures of 500 to 600 psi have been recorded.

The mixing vat, connected to the pump suction by a suction hose, is a cylindrical tank 3 ft diam by 30 in. high, fitted with a Gardner-Denver air motor directly connected to a paddle arm suspended vertically in the center of the mixing unit. A metal screen is fitted across half the mixing tank for screening out lumpy cement.

The only other items of necessity besides the air and water connections, several 2-in. gate valves, 1½-in. pipe, and line oilers are a small handcar for carrying cement, two lengths of high pressure hose for use from discharge of pump to 2-in. valve or packer and a bypass arrangement from the pump

The 7x3x10 pump was torn down daily for cleanup. Note rubber pistons in end of liquid cylinder. Valve on drift rib marks completed hole.

discharge back to the mixing tank. The best high pressure hose that has been found to date is a standard Eimco, wire braid, 2-in. air hose. It is also advantageous to have at hand three packers for use under emergency conditions or for temporary stopping of smaller flows of water.

As a matter of convenience and practical operation the grouter unit should be as close to the working face as possible, although grout has been pumped from a distance of over 200 ft without any particular difficulties. Prior to actual grouting the hole should be allowed to run freely at several different intervals in order to flush out as much of the silt, sand, and crushed material as possible.

Oftimes, after the valve has been closed on a newly struck water zone, the face of the heading resembles the lava cliffs at Thousand Springs, Idaho. The pumping of one, repeat *one*, small handful of fine sawdust will usually seal off all seams sufficiently enough to hold the cement mixture. Bitter experience showed that a little sawdust will do a lot of good, but a lot of sawdust ruins the hole as well as clogging the pump. The small amount of sawdust required is one of the difficult items to teach a green operator and usually can be learned only by actual experience.

It has been found good practice to pump several hundred gal of clear water into the hole prior to the grout mixture. This procedure seems to clear any possible obstructions, and if colored water is used, can also serve as a tracer for possible grout return elsewhere in the mine.

Many industrial grout operators favor the use of a second mixing vat in order to obtain a certain cement-water ratio. When the grout program was first initiated a second tank and mixer was not immediately available so that work started by mixing batch units in a single mixer tank, pumping it out and then remixing another batch. This, obviously too slow a process, led to experiments with a continuous batch procedure governed by the experience of the individual grout-pump operator. This is the method today, as it was found the operator quickly became adept at gaging the consistency of the mixture and was able to immediately adapt the grout unit to any change in pumping conditions, adding more cement for loose pumping and thinning down the mixture for tight. Average conditions at Deep Creek require from 2 to 3 sacks of cement per tankful of water, which approximates 40 to 30 gal of water per sack. On occasion less than 1 sack and as

LEFT: Grout mixing tank has air motor suspended vertically in center. Suction pipe is being positioned. CENTER: Alex Koski inserts packer unit in hole. Tightening nut expands rubber washers at end of packer to cut off flow and secure unit. RIGHT: Hardened cement filling 4 to 6-in. fracture is exposed by 650-level drift. Water pressure here was 250 psi while grouting.

high as 6 sacks per tank were used. Cement placed in one hole has varied from a low of 1 sack to a high of 1500 sacks.

Naturally, once pumping has begun, it should be continued on a 24-hr basis until the hole has been completely plugged. However, underground, this procedure is not always feasible due to blasting in the near vicinity, the lack of trained personnel to operate the three shifts, or for some other good reason. It has been noted that the intake of the hole does not seem to be greatly impaired if two or three tanksful of clear water are pumped in immediately prior to going off one shift, three or four tanksful of water pumped intermittently during the following 8 hr with a second operator taking over on the third shift.

Pumping must be continued to complete refusal. If properly grouted, the hole will be solid cement to the end of the pipe insert or packer. The 2-in. valves and packers must be removed within 3 hr after completion and thoroughly cleaned of all cement. It is usually a good idea to drive a wooden plug into each hole from which a valve or packer has been taken as a precaution against losing any of the green cement. Inasmuch as the bulk of the grout is pumped against relatively high pressures it has been found to be good practice to thoroughly clean, once per 8-hr shift, all of the valves, seats, and liquid chambers of the pump, at the same time inspecting and replacing, as required, worn valve seats, valve inserts, and pistons. The same clean-up process must be performed immediately after completion of the hole, for with the high pressures obtained at final refusal the grout still existent in the pumps and lines will set-up with great rapidity.

Refusal to pump prior to actual finishing of hole can usually be traced to freezing of the pump exhaust, a problem for which no satisfactory solution has been found to date, or clogged valve ports. Operation of pump without apparent injection of grout is usually indicative of faulty valve seats or of piston failure.

It would be unnatural if special problems did not arise on occasion. In one particular incidence a crosscut had intersected an unexpected fault whose strike was roughly parallel to the line of the tunnel. The large volume of water encountered under approximately 200-lb pressure washed out an opening 5 ft by 14 in. for a distance of 8 to 10 ft up the dip of the fault plane. Two, 20-ft longholes were drilled which tapped the water flow above and approximately 15 ft from the open vent. A 3-in. plank bulkhead, lined with burlap, was wedged and blocked into place in the fault opening, a length of 1½-in. pipe extending through the plank for a distance of 5 ft into the washed out section. The grouter was hooked on and a thick slurry of cement and sawdust was pumped through the bulkhead until a return was noted in the reliever holes. Pipes and valves having already been installed in the two reliever holes, grouting was then continued through them until completion.

On another and much more spectacular occasion the bottom of one of the drifts suddenly blossomed out with an artesian spring whose flow of 500 gpm caused some frantic activity. Three holes were drilled in the bottom of the drift about 15 ft away from the *gusher* hole, the third one finally intersecting the water channel about 15 ft below the drift. A plug of wood and burlap was wedged into the spouting hole and an ordinary drill column was mounted on top of the plug and screwed down tight. A thick mixture of grout and sawdust was pumped into the reliever hole and after setting up a sufficient length of time the grouting was finished and the water completely sealed off.

A predetermined cost of grouting is impossible to attain, nor would it be of any particular value for comparison of one mine with another. In this case, grouting has graduated from being a specialty item and has become a vital part of regular mining procedure. Plans call for the permanent exclusion of 60 to 70 pct of the present flow of 800 gpm, the latter representing a drop of 50 pct within the last 2 years as well as the elimination of water tapped by development during the same period of time.

It is felt that the cost to date of approximately 8 pct of total mine operating expenditure, high enough in total, is but a relatively small price to pay for the tonnage made available for past and future extraction. After the present grouting program is completed the continuing expense should not exceed 2 pct.

Looking back over this experience the author feels that the mining industry has not taken enough advantage of this weapon against high pumping costs. When a sudden water problem raises its head one is too prone to call for additional pumping facilities, a costly and perpetual expense. Perhaps many thousands of dollars could be saved if consideration were given to the possibility of settling the problem by eliminating it—permanently. Substituting for the words of one of the national advertisers, perhaps the industry slogan should be: "Next time—try a grout pump!"

Stress Concentration Problems in Hollow Drill Steel

by W. H. McCormick and H. J. Benecki

CONSIDERING that a typical modern drilling machine strikes approximately 2000 blows per min and may develop a force of 30 to 200 ft-lb, depending upon size of the drill used, the importance of uniform stress distribution in this type of drilling becomes obvious. If the rod were perfectly symmetrical, and if it did not rotate, the problem would be relatively simple, however, the necessary forging, machining and heat treating operations prevent attaining this state of perfection.

Take the modern hollow drill rod. The blow is transmitted not from the striking end of the rod directly through to the point of impact of the bit in the rock, but rather through a somewhat complex tool. To add further problems, this entire assembly is rotated immediately after the impacting blow. Rather than attempt to discuss the magnitude and distribution of stresses, it is desired to present a few things that will minimize unusual stress concentration and produce a rod in which the stress distribution will approach the theoretical ideal.

Briefly, stresses to be considered are:

(1) When the piston or anvil block strikes the end of the rod, it is obviously put under compression.
(2) When it rebounds, it is equally obvious that the entire rod is in tension.
(3) Rotating the rod after each blow introduces torsional stresses.

Inasmuch as any one individual force applied is below the yield strength of the rod, failures invariably occur because the fatigue strength of the rod has been exceeded. A commonly accepted definition of fatigue strength is the strength which the material will withstand in repeated stress for an infinite number of cycles. Therefore, if the hollow drill rod were perfectly made and used under perfect conditions, it would last indefinitely.

Modern laboratory fatigue testing is conducted with precisely machined specimens. It is difficult to correlate such results with actual field fatigue life. The block test has been developed to simulate actual service conditions, and these results when correlated with actual rock test results provide a criterion for hollow drill steel.

W. H. McCORMICK and H. J. BENECKI are Chief Metallurgist and Metallurgist, Park Works, Crucible Steel Co. of America, at Pittsburgh.

Bit and shank design are important. Over a period of many years the integral forged bit was developed to its present shape, and gives satisfactory results. Design of detachable bits has progressed to such a point that they also provide excellent performance in most instances. The integral carbide rod represents the latest cutting end to be developed. But, this type of rod is not made by merely upsetting a piece of hollow drill steel, slotting it and inserting a piece of carbide. It must be designed to minimize stress concentration.

Shanks have been standardized in design and when properly made will provide a maximum of service life. Unfortunately, they are not always properly made. An ample fillet is always a must when collared shanks are made. An abrupt change in section provides a point of stress concentration and failure would occur almost immediately. The pear-shaped collar has been developed to minimize this condition.

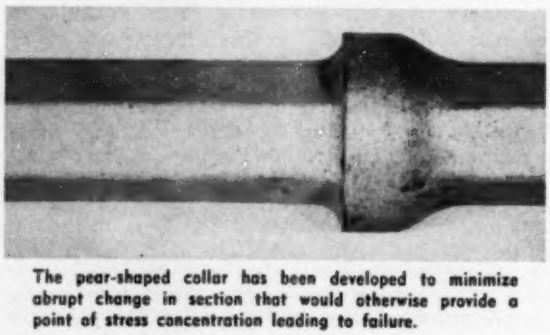
All are aware of the importance of preventing forging laps. Since there continue to be rods that have failed prematurely because of this, the importance of proper pinning in shank making should again be emphasized.

The sectioned shank illustrated is a case where too much metal was allowed to gather in the hole prior to pinning. When the pin was inserted instead of expanding the metal a lap was formed. Stresses will concentrate at the bottom of this lap and early failure can be expected.

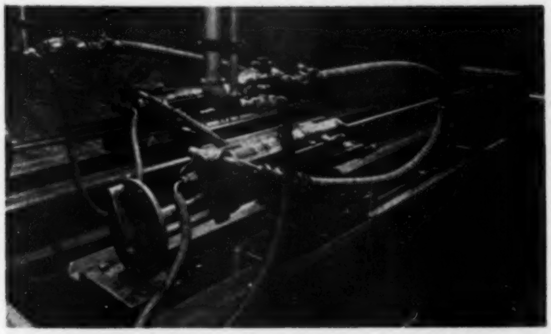
Also shown is a collar which, although it has a proper fillet, will fail prematurely because of the presence of a lap. Invariably, this is caused by excess metal in the die, worn dies or overheating.

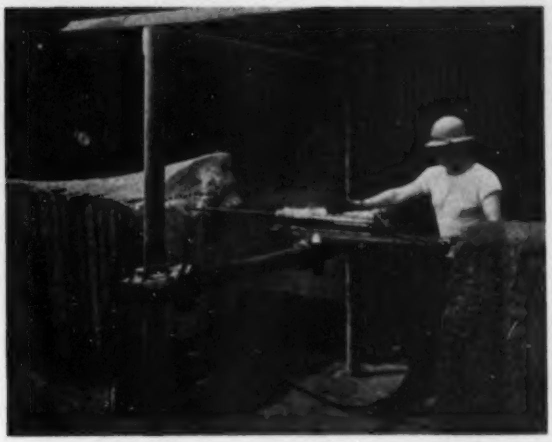
Bits available today, whether of the one-use or multiple use type are very carefully designed to provide a good fit so that there is no problem if the bit manufacturer's instructions are followed carefully. To insure this, attention must be paid to the

Forging laps are one cause of premature failure. Sectioned shank at left shows lap from improper pinning, and collar at right has lap due to improper upsetting.



The pear-shaped collar has been developed to minimize abrupt change in section that would otherwise provide a point of stress concentration leading to failure.





For drill steel development, testing must go beyond laboratory fatigue testing with machined specimens. Block test simulates service conditions, above. Results are checked with actual rock tests, as at right.

proper maintenance of the threading equipment. Even with the best bit on the market, thread failure will be encountered if there is a sloppy fit.

With the trend toward greater use of alloy rods, the scaling characteristics must be considered to prevent excessive scaling and subsequent undersize threads. Mr. W. W. Durand, in a recent article in *Mining Congress Journal*, discusses this very thoroughly and shows the importance of proper heating and proper furnace atmosphere.

In recent years a lot has been said about the importance of prestressing. The theory is that by putting surface layers of the metal under compression by cold working, the fatigue life will be materially extended. This is an established practice in many applications today and is well illustrated in the case of threaded rods by the use of the Ingersoll-Rand Jackroll. The object of this tool is to cold work, or prestress, the thread undercut, which is always a zone of stress concentration even on very well made attachment ends. If the stress raising effect is too severe, premature failure is a natural result. It is the authors' belief that, if practical, the entire thread should be similarly prestressed. The advantage of prestressing extended to the entire rod is certainly a step in the right direction. The Gardner-Denver Co. has shown excellent improvement in fatigue life by shot peening the entire rod.

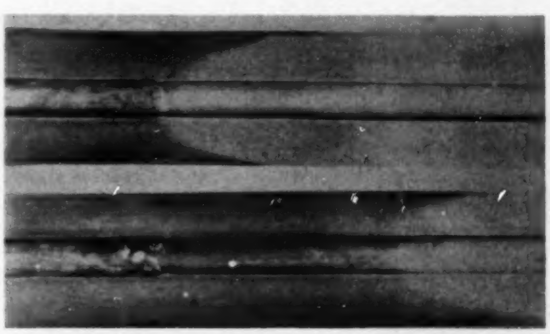
So far we have discussed things of a mechanical nature which will cause disastrous stress concentration areas. The metallurgical notch, which again has been discussed often, must still be kept in mind. Failures are still encountered due to not observing the necessity of preventing heat treating zone concentration. This condition may be found in either the attachment end or the shank end if they are not properly heat treated. By proper heat treating we mean following the manufacturer's recommendations of carefully overlapping the heats prior to quenching.

The metallurgical notch is just as serious as that caused by the forging lap and must be prevented. It is wrong to think that the alloy rods are any different in this respect from the carbon type. In both, the abrupt change from heat treated to natural steel must be prevented.

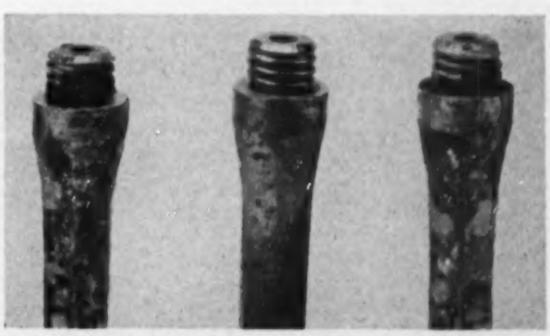
Abuse causes marks at which stresses are concentrated. This has been mentioned frequently but inasmuch as it still causes rod failure, it is worthy of repetition. Just a simple thing like using a chisel to open a bundle of rods, or a badly scuffed up chain to handle them, can produce premature failures. Although the alloy rods are generally harder, if anything, they are more susceptible to early failure than the carbon rods if handling nicks, tool marks, and the like are present.

The hollow drill rod, by its very nature, can never be of the best design to accommodate the many stresses encountered in this type of drilling. It becomes evident therefore that every precaution possible must be observed to eliminate the zones of stress concentration. With the continuing growth of the alloy drill rod it becomes apparent that modern forging and heat treating facilities are a must.

The Climax Molybdenum Co. in an excellent booklet, *Three Keys to Satisfaction*, summarizes very nicely what is discussed here when it states that, "good design plus good steel plus good treatment equals satisfaction."



Comparison of properly overlapped and improperly treated rod ends shows one cause of early rod failure.



Undersize thread ends, left and right, result if scaling characteristics are not considered in heat treatment.

Open Pit Forum

Kaiser's Jamaican Bauxite Operation

by A. L. Moore

Typical operation of truck-shovel pit at Kaiser's bauxite mining area, New Buildings mine, southwest Jamaica.

KAISER Bauxite Co., a subsidiary of Kaiser Aluminum & Chemical Corp., has been mining and shipping Jamaican bauxite for over a year. On Feb. 10, 1953 the first boat load of bauxite left Port Kaiser, Jamaica, for the company's alumina plant at Baton Rouge, La.

To provide its own source of the aluminum ore, a Kaiser Aluminum & Chemical Corp. exploration party started initial reconnaissance in 1947. This work, and subsequent exploration, led to the acquisition of reserves sufficient to warrant a full-scale mining operation for several decades.

The design and construction of the port facilities, railroad, and staff camps was performed by Kaiser Engineers, a division of Henry J. Kaiser Co. The administrative offices are at Spur Tree, the open pit mining operation is in St. Elizabeth Parish, and the stockpiling, drying, and loading facilities at Port Kaiser. A standard gage railroad transports ore from mine to the port area.

Present mining is at New Buildings, St. Elizabeth Parish. The surface topography of the 850 acres consists of coraliferous limestone hillocks, separated by gently sloping depressions. Exposed limestone is thickly vegetated, and the depressions contain the soil areas which form the bauxite deposits.

Ore Occurrence

Ore deposits occur directly at the surface, which simplifies mining by open pit methods. Although there is no overburden as such, Jamaican mining regulations require that the first 6 in. of topsoil be removed and stockpiled prior to mining. After mining has been completed, the topsoil is spread in the mined-out pit to return the area to its former productivity.

The orebodies at New Buildings are bowl-shaped deposits underlaid by limestone. It is generally accepted that the bauxite was formed by weathering of the limestone.

Areas which will be initially developed vary in size from an acre to as much as 20 acres. Ore thickness ranges from 1 to 107 ft, with an average depth of 20 ft, with 5 ft considered as minimum mining depth. The deposits are irregularly scattered over an area of 455 acres.

Haulage

To link the deposits with the loading ramp at the rail head, 4 miles of main haul road have been constructed. The roads have a minimum width of 25 ft, and are crowned, superelevated, and adequately drained. Maximum grade against the load is 3.5 pct, and all curves have adequate radius. Roads are surfaced with 4 in. of crushed limestone, and maintenance is accomplished by a motor grader and a 10-ton roller. To allay dusting, the loading ramp, parking areas, and main haul roads are treated with oil. In dry periods, a sprinkler truck is employed on the feeder roads and pit ramps.

To connect the mining locations to the main haul roads, ramps and feeder roads are constructed as required. To avoid ore dilution, these subsidiary roads and ramps are not surfaced when located in bauxite zones.

Shuttle haulage of ore at New Buildings is accomplished by a fleet of model 80 F. D., 15-ton, rear-dump Euclids. The capacity of the trucks has been increased by the addition of 18-in. sideboards. The broken bauxite averages 75 to 80 lb per cu ft, which allows an increase in the volume carried by

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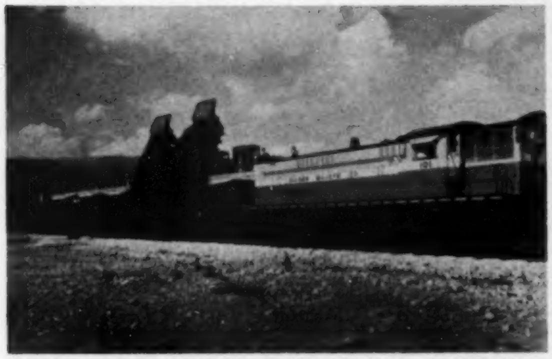
View of Port Kaiser shows 450-ft long pier, built with combination of open approach and rock-fill. Dredging was not necessary because of depth of water off point. Maintenance shop, warehouse, and office also located at the port.



Ore train being loaded at the New Buildings ramp for the 12.5-mile trip to Port Kaiser. Two Baldwin 1200-hp diesel-electrics handle the trains.

Rotary car dumper at Port Kaiser handles incoming 70-ton gondolas. Portion of the boom stacker and stockpile shows in background at left.

the haul units without exceeding their rated payload. In dry periods, the moisture content of the raw bauxite averages 18 to 19 pct. When the moisture is increased by rainfall the ore is sticky, and to assure clean-dumping the Euclids have heated beds.

A railroad loading ramp constructed of precast concrete slabs and columns is located at the center of the mining areas at an elevation to allow the Euclids to dump ore directly into railroad cars. A construction feature of the ramp is that it can be torn down and reassembled at another site.

Ahead of mining, the deposits are drilled on 50-ft centers. The drilling is done manually, using sectional drill steel with detachable augers. The cost of hand drilling compares favorably with that of machine drilling, and has the added advantage of being more flexible in rough terrain. From drilling results the shape of the deposit is defined and ore estimates are made. Bottom-of-ore contours and cross-sections are plotted to aid pit operation. Due to the uniformity of the ore, it is sufficient to sample alternate holes on the 50-ft grid to obtain a satisfactory grade analysis for the orebody.

Mining

Before an area is mined it is cleared by bulldozer, then approximately 1 ft of the topsoil is removed by scraper and stocked outside the ore limit. In mining the larger deposits, the usual procedure is a cut or cuts by shovel across the entire area; access ramps being constructed as the mining depth increases. The contact between the bauxite and underlying limestone is sharp, but highly irregular. After shovel operation is no longer feasible, a dragline mines the larger remaining sinks and pipes of ore. The smaller, steep-sided orebodies are mined entirely by dragline.

Three deposits are mined simultaneously. For digging and loading the ore, one Marion 93M shovel-dragline, one Marion 111M shovel-dragline, and one Marion 111M dragline are used. For general cleanup a bulldozer is assigned to each pit.

Experimental blasting to break the ore was conducted in the first pit to be worked. Due to the highly porous nature of the bauxite, the breakage obtained was not commensurate with the drilling and explosives cost. To aid excavation in zones of hard ore, a tractor with rooter attachment works ahead of the loading unit.

Pit drainage is not a major problem. Due to the porosity of the bauxite and the limestone, water collects in the pits for short periods only after prolonged and heavy rainfall. Two rainy seasons are usually encountered each year, occurring in May and October. During the periods of heaviest rainfall, haulage over the unsurfaced ramps and feeder roads becomes difficult. To prevent interruption of ore shipments, a suitable stockpile is maintained adjacent to the loading ramp as well as at Port Kaiser.

Surface facilities at the New Buildings mine consist of a field maintenance shop, warehouse, and space for equipment parking. The shop building houses three bays for tractor and Euclid maintenance and servicing, as well as office space for the mining superintendent and clerical help. Major overhaul and repair is accomplished at Port Kaiser.

Railroad

The single-track, standard-gage railroad is 12.5 miles long. The ruling grades are 2.2 pct with the load and 1.5 pct against the load. All grades are compensated for curvature, and maximum curvature on the main track is 10°.

In benching the cliff section on the final approach into Port Kaiser a local fault zone was encountered. To render this section safe for railroad operation, it was necessary to loosen 345,000 cu yd of limestone with a coyote blast. The blast used 110 tons of dynamite.

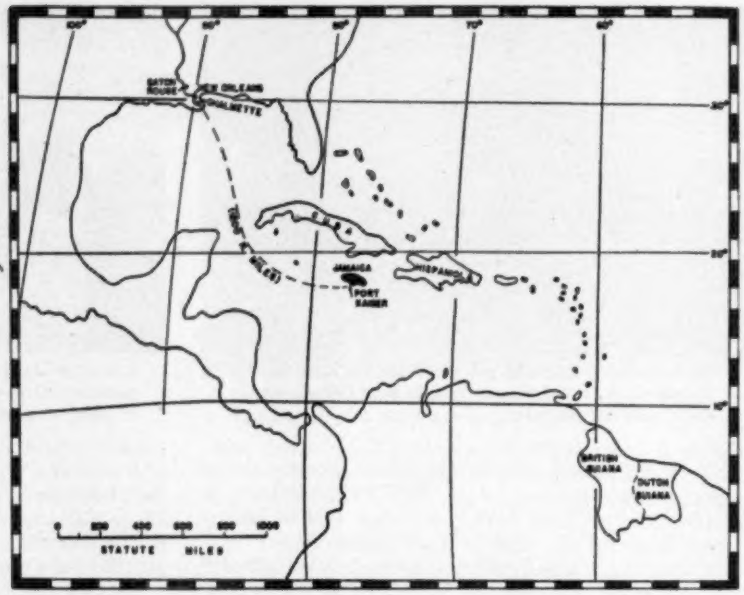
For main-line ore haulage, two Baldwin 1200-hp diesel-electric locomotives, and standard, 70-ton gondola cars are used. The locomotives operate in either direction, and are equipped with dynamic braking. Additional rolling stock includes a 65-ton diesel-electric switcher at Port Kaiser.

Each locomotive makes four round trips daily between New Buildings and Port Kaiser. One string of cars is loaded at the mine, while a second string is being dumped at Port Kaiser, and a loaded train and an empty train are in transit. To aid train movement, a telephone system connects the mine, the passing track, and the pier area.

Port Facilities

At Port Kaiser on the south coast of Jamaica are located the facilities for drying, storing, and shipping the bauxite mined in St. Elizabeth Parish. As the ore trains arrive from the mine, the cars are switched and dumped singly through a rotary car dumper. A reinforced concrete hopper under the car dumper feeds to five, 72-in. apron feeders. The feeders are independently motored, and operate normal to the center line of the car dumper. The apron feeders discharge to a 42-in. inclined belt conveyor which transports the ore to a stationary boom stacker. The stacker has a 120-ft movable

Map, right, shows the sea route from Port Kaiser to Baton Rouge. This route cuts ore haul 60 pct as compared to trip from the Guianas. Sketch above shows location of Kaiser operations on the southwest coast of Jamaica.



boom which carries a 42-in. belt conveyor, and can rotate through a 39° stacking swing. Space is provided for increasing the capacity of the stacker stockpile by bulldozing.

Two 2-cu yd crawler-mounted clamshells reclaim ore from the stockpile into the dryer hoppers.

The two, direct-heat, cylindrical dryers are 10x 80 ft. A refractory-lined, steel combustion chamber burning Bunker C fuel oil is provided for each dryer and the heated gases are passed through by draft fans. A multiclone dust collector is located between each dryer and draft fan.

Raw ore is fed from the dryer hoppers to the dryers by pan feeders. At the present time the bauxite is being dried to 15 pct moisture. The dried material discharges at a temperature of about 130°F to 30-in. belt conveyors at right angles to the dryer cylinders. From these belts the ore passes to a 42-in. inclined belt conveyor leading up and into the dry storage building.

Dry Ore Handling and Loading

The dry storage building is a fully enclosed semicircular section structure of steel trusses covered with corrugated aluminum sheet. Dried bauxite is distributed throughout the length of the building from a height of 50 ft by a traveling tripper. Stored ore is reclaimed through ten gates located in the roof of a reinforced-concrete tunnel which runs lengthwise under the storage building. The gates are air-operated and draw to two movable pan feeders astride the 42-in., reclaim conveyor belt which moves the ore to a transfer station west of the storage building. From this transfer point, a 42-in. belt runs to a transfer station at the pier head. An automatic sampler and a conveyor scale are located at the pier transfer station.

Bauxite from the pier conveyor belt goes by a traveling tripper to a short cross conveyor and discharges to the inclined trailer conveyor of the self-propelled gantry ship loader. The gantry boom carries a 42-in. belt conveyor which empties into a 32-ft telescoping chute suspended from the boom. A ship trimmer at the base of the chute was found unnecessary to properly distribute the bauxite in the holds of the ore carriers.

All controls for the movement of the gantry and its trailer conveyor, retraction or suspension of the gantry boom, and telescoping of the boom chute are located on the deck of the gantry.

In addition to the dry storage building, the major facilities at Port Kaiser consist of a maintenance shop, warehouse, and office building. The maintenance shop contains space for the overhaul and repair of all equipment used in the mining, transporting, and loading-out operations.

Two 50,000 gal, and two 250,000 gal tanks allow storage for diesel and Bunker C fuel oil. Fresh water for the port area is brought in by rail from the deep well at Duff House, and is stored in two 8000 gal underground tanks.

Staff Camps

At Spur Tree, 6½ miles from the town of Mandeville, are the administrative offices, laboratory, and staff housing for the American personnel. The area has its own power plant, and water is obtained from a 1000-ft well located in the valley below the campsite. At an elevation of 2000 ft, the camp enjoys a mild climate. The nights are cool, and the daytime temperature rarely exceeds 80°F.

Mandeville is connected with Kingston, the capital and principal harbor of Jamaica, by 66 miles of hard-surfaced road.

Quarters for native foremen and technicians are maintained at Duff House. Bungalow and apartment-type housing with modern facilities is provided. A 550-ft well supplies water for the camp, and is also the source of fresh water used at Port Kaiser. Power is furnished by a line from the Port Kaiser plant. An access road, paralleling the railroad, connects this camp with the port area.

The inauguration of ore shipments from Jamaica gives Kaiser a well-integrated aluminum operation. The ore boats from Port Kaiser unload directly at the pier of the Baton Rouge alumina plant located on the Mississippi River. Approximately half the alumina produced at Baton Rouge will be reduced to metallic aluminum at the company's Chalmette reduction plant 86 miles to the south. The balance is required at the Tacoma and Mead, Wash., plants.

A Realistic Look at Taconite Estimates

by John W. Gruner

On account of the tremendous increase in the production of steel there has been much speculation as to the reserves of iron ore. A number of estimates of world scope have been published, which show that these reserves are definitely limited if a long range view is taken. There are, however, optimists who believe that in perhaps two generations the metallurgy of iron will have advanced to the point where any kind of iron mineral combination of perhaps 20 pct iron will be usable. This thinking has been particularly applied to the new taconite developments of the Mesabi Range, for here is an iron formation of such vast extent that its iron could almost last forever.

Let us briefly examine this most favorable of all low grade ore reserves in North America. Nothing will be said about the 1,000 million long tons of material now classified as ore, for they will be largely gone in less than 20 years.

The word *taconite*[1, 4] is used by geologists to designate the rock which makes up the Biwabik iron formation that outcrops along a belt about 100 miles long. It is from 400 to 750 ft thick and dips with some exceptions at 5 to about 12° southward. The dip determines the width of the belt of outcrop under the glacial drift and at maximum is about 2½ miles wide. Taconite contains on an average 27 pct Fe and 51 pct Si, but ranges from about 15 to 40 pct Fe with corresponding changes in silica.

The tonnage of this rock is enormous. The writer has calculated that between the towns of Mesaba and Grand Rapids, a distance of 71 miles, 89 billion tons underlie the outcrop assuming a vertical cut-off at the southern edge of the formation.

The East Mesabi Range,[3] a 14-mile eastern extension, is metamorphosed by the intrusion of the so-called Duluth gabbro. This taconite would add about another 10 billion tons but for mineralogical reasons it should be kept separate. The extension west of Grand Rapids being little known will be left out of all our estimates. These 89 billion tons contain about 24 billion tons of iron. There are, of course, very much larger quantities stored down the dip to unknown depths. The bulk of the iron in the taconite is chemically combined in the form of iron silicates which contain an average of 45 pct SiO_2. Since the taconite is very low in lime, it possesses none of the self-fluxing properties that, for example, are found in the Minette ores of France which contain 12 to 15 pct SiO_2. It is, therefore, unrealistic to believe that a metallurgical process might be found that could cope with the high-silica of the taconite.

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The only possibility then is that portions of the taconite can be separated by mechanical means so that they give a usable concentrate. This is where the magnetite-bearing taconite comes in. Beds and lenses of this material occur at certain horizons in sufficient thicknesses and concentrations so that quarrying has been undertaken on a pilot plant scale at three places and should in 3 more years be in large scale production. This magnetic material in the beds which are considered workable averages about 22 pct Fe in the form of magnetite. A little more than 3 tons of it will make 1 ton of high grade concentrate of over 60 pct Fe and 10 to 12 pct SiO_2. Since all of the minerals including the magnetite, quartz, iron silicates, and siderite are very fine-grained, the problem of obtaining a satisfactory concentrate kept specialists busy for many years.

The available tonnage of the magnetic layers must be divided into those which can be quarried by open pit methods and others which are accessible only by underground mining. The former are the only ones that are in the picture today.

The author estimated in 1946[2] on the basis of drill samples that magnetite could be quarried to a depth of about 230 ft in large areas and would average a thickness of 100 ft in the so-called Lower Cherty division of the iron formation. The glacial drift is not included in this depth and must be considered overburden. These magnetites extend from the east end of the range to about Nashwauk, a distance of 60 miles excluding the hematite-limonite ore areas. Each running mile to this depth should average 75 to 90 million tons of magnetite taconite. The total tonnage, therefore, is estimated at 5 billion. An optimistic estimate might add another billion for some very fine-grained magnetic material in the central part of the range which until lately had been considered beyond recovery. These taconites would yield about 2 billion tons of concentrates obtainable relatively easily compared with material which could be mined only by underground methods.

If it should ever be economically possible to produce the magnetic taconite from underground, the

MINNESOTA TACONITE ESTIMATE

Reserves which can be extracted magnetically

Formation	Reserves
Open pit including East Mesabi Range	5 to 6 billion tons
Underground from Lower Cherty division 260 million tons per square mile area.	
Total for 1 mile width to depths of 800 to 1500 ft depending on local dip of formation.	21 billion tons
Additional tonnages from Upper Cherty division for a belt of the same width below zone of alteration.	10 billion tons

tonnages become enormous. The same layers of the Lower Cherty division mentioned above if followed *underground* contain about 263 million tons of magnetic taconite under every square mile. For a length of 80 miles and a width of 1 mile this is equal to 21 billion tons. The deepest workings would have to be 800 to 1500 ft depending on the dips, which would be 5 to 14°. Every additional mile down the dip should contain comparable tonnages. There are also magnetic taconites in the so-called Upper Cherty division. With exception of the East Mesabi Range, they are largely altered to hematite in most places within quarrying depth. Below the oxidized areas, that is at depth out of open pit range, the Upper Cherty division would contain almost 10 billion tons between Mesaba and Nashwauk for a belt one mile wide as measured down the dip. These magnetites would be the least desirable from the standpoint of availability.

Nonmagnetic taconites chiefly because of chemical combination of iron with silica cannot be considered reserves by any stretch of imagination. A typical mineralogical analysis of this material is 22 pct quartz, 68 pct silicates and 10 pct iron carbonate. It contains, in other words, about 32 pct combined and 22 pct free SiO_2 neither of which can be separated by known metallurgical methods.

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[2] J. W. Gruner: Mineralogy and Geology of the Mesabi Range: Iron Range Resources & Rehabilitation Commission, State Office Bldg., St. Paul, Minn. (1946) 127 pp.

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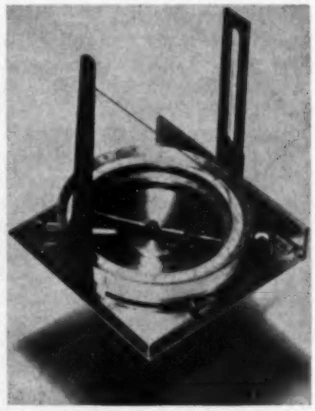
The Engineer's Notebook

Dial Compass For Exploration



The dial compass first appeared in the 15th century. First reported use for exploration was by T. B. Brooks and R. D. Irving who designed the present form of the instrument for their work on the Michigan iron ranges. Instrument shown is from W. & L. E. Gurley Co.

ENGINEERS and geologists find the magnetic compass unreliable in many mineral bearing areas because of local magnetic attraction. This is the case in the vicinity of many contact deposits, and particularly true in the Lake Superior district where iron formation is wide spread and causes wide fluctuations of the magnetic needle. One of the most useful instruments for geologic traversing or running grid and claim lines under these conditions is the dial compass which does not depend upon magnetism.

Difficulties encountered in the field because of local attraction can be overcome through transit observation of Polaris or the sun, or use of the solar attachment. However, a field party often lacks the equipment or time for work of such accuracy and it is here that the dial compass finds its use.

The dial part of the instrument operates on the same principle as a sun dial, and has an hour circle graduated in divisions of 5 min of time. A watch marks uniform time intervals, but time indicated by a sun dial coincides with *watch time* only four times a year. During the remainder of the year the sun dial is ahead or behind watch time by a variable amount given by the *equation of time*. Watch time is changed to dial compass time by applying an overall time factor that includes correction for longitude, the equation of time, instrument error, and the latitude design of the instrument. The equation of time for any day can be obtained from a Solar Ephemeris or Nautical Almanac, (obtainable from the Supt. of Documents, U.S.G.P.O., Washington, D. C.).

Field procedure for measuring overall time correction involves orientation of the compass on a true meridian, recording the difference between apparent or dial compass time and watch time for different times of day, and making up a correction table. This correction table or chart will not be correct for a full summer season because of the daily change in the equation of time, but this change can be obtained from a Solar Ephemeris.

In traversing for geologic mapping the instrument is used to maintain the line of travel. The compass is leveled, rotated until the shadow of the thread shows watch time, plus or minus the overall correction from the table previously made up, and the north-south direction is sighted in. With a little experience an operator can often travel faster than with a Brunton. Greater accuracy can be obtained by mounting the instrument on a Jacob's staff, although it is often hand held. For laying out grid lines the compass is normally mounted, as it is when used to orient a plane table for true bearing.

A true meridian for making the initial time correction table can be established from known survey points, by sighting distant points where an accurate map is available, and can most accurately be obtained by observation of Polaris, following methods given in surveying texts. Accurate watch time is a must. Watches may be checked during long periods in the field by making a setup for observation of a particular star, noting that the star will set 3.93 min earlier each night. Another procedure is to check back from a meridian, using the dial compass time. *(With portable short-wave radios becoming more available, observatory time signals can also be used for checking watches. Ed.)*

Simple to operate, fast, and reasonably accurate, the dial compass is a tool that should not be overlooked by field parties operating where local magnetic attraction is known or suspected.

This article was abstracted from a paper by Professor D. H. YARDLEY of the University of Minnesota.

Upgrading Manganese Ore

Three Kids Mine, Nevada

by S. J. McCarroll

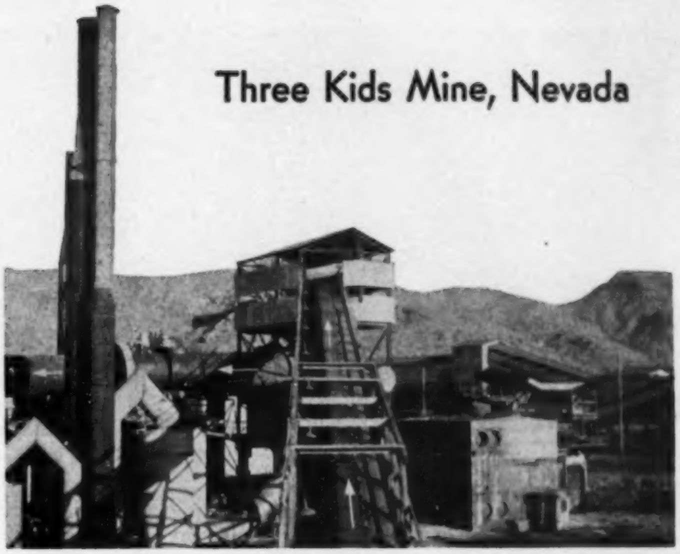


Fig. 1—The belt shown at right carries filter cake to mixing station over calciner. Crude ore conveyors appear in right background.

THE Three Kids mine, some six miles east of Henderson, Nev., is in a typical southwest desert area, with high dry summer heat and cool to cold winter seasons.

The manganese deposit was located during World War I.[1] During this period 15,000 to 20,000 tons of ore assaying up to 41 pct manganese were shipped. Interest in the deposit was not revived until the middle thirties, when experiments on the ore were initiated. Test work indicated possible recovery of only 70 pct by flotation, but in 1941 additional work was done at the Boulder City pilot plant of the U. S. Bureau of Mines and also by M. A. Hanna Co. As a result, the Manganese Ore Co. was formed and a plant utilizing the SO_2 process was constructed. Numerous operation difficulties ensued, and the plant was closed when the manganese situation in the country eased.

In 1949 Hewitt S. West initiated negotiations to acquire the plant. In 1951 Manganese, Inc., was formed and contract entered into with the General Services Administration to supply 27 million units of metallurgical grade manganese in the form of nodules to the national stockpile. A second contract was made to upgrade 285,000 tons of stockpile ore.

Test work was undertaken by the Southwestern Engineering Co. and likewise by the Boulder City pilot plant at the U. S. Bureau of Mines. Results obtained indicated the commercial feasibility of the flotation process. Construction of the plant, which is shown in Figs. 1 and 2, was started in June 1951, and operations on a break-in basis began in September 1952. Apart from the usual starting difficulties two major disasters caused serious setbacks, one a kiln failure in February 1953, and the other a fire that destroyed the flotation building in June of the same year. The nodulizing section of the plant resumed operation in November, and the flotation section in January 1954.

The ore minerals are chiefly wad,[2] with minor amounts of psilomelane, and occur in sedimentary beds of volcanic tuff. The ore is overlain with beds of gypsum which outcrop or may be covered with surface gravel. Intermediate beds of red and white tuff occur frequently with lenses of red and green jasper and stringers of gypsum and calcite. Small amounts of iron are present; lead content averages about 1.0 pct and minute amounts of copper and zinc are found. Barite, celestite, and bentonite are present. Since these are made up of minute asicular crystals, moisture content is very high, averaging about 18 pct. Ore reserves have been estimated at 3 million tons averaging 18 pct Mn[2] and up to 5 million tons after grade is dropped to 10 pct Mn.

A good part of the orebody was stripped of overburden by the previous operating company. Approximately 50 pct of the ore, representing more than 60 pct of the manganese, can be mined by open-cut methods. A system for underground mining has not yet been decided on.

Open-cut mining with benches of 20 ft has proved satisfactory. Although the ore is soft and appears dry and dusty it has a certain resilience, probably due to the porosity and moisture which makes drilling and fragmentation difficult. Wagon drills have been abandoned in favor of the Joy 225-A rotary drill which will put down a 4¾-in. hole at the rate of 2 ft per min. Holes are spaced in a pattern with 8 to 9-ft centers. Forty percent powder has been used, but better breaking to 2-ft size is obtained with low velocity bag powder of 30 pct strength.

Loading is done with one 2½-yd shovel, and cleanup follows with one D-7 bulldozer. The ore is hauled with Euclid trucks about 1000 ft from the pit to a blending pile, where the daily mine production is spread in layers by bulldozing until approximately one month's mill feed is accumulated. A new pile is then started and mill feed is drawn from the first pile by one 1¾-yd shovel and Euclid trucks, with a haul of approximately 500 ft.

Mining is performed by an independent contractor with engineering and supervision by the company staff.

Early test work indicated that the manganese could be floated with soap, a wetting agent, and fuel oil to give a recovery of better than 75 pct with a grade of 43 pct Mn. The concentrate when nodulized with coke would upgrade to 46 pct Mn or over, and the lead volatilized to 0.6 pct residual.

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Discussion on this paper, TP 3747B, may be sent (2 copies) to AIME before May 31, 1954. Manuscript, Sept. 8, 1953. New York Meeting, February 1954.

The Transactions papers appearing in Mining Engineering during 1954 will subsequently appear in AIME Transactions Vol. 199 and may be permanently referenced to that volume.

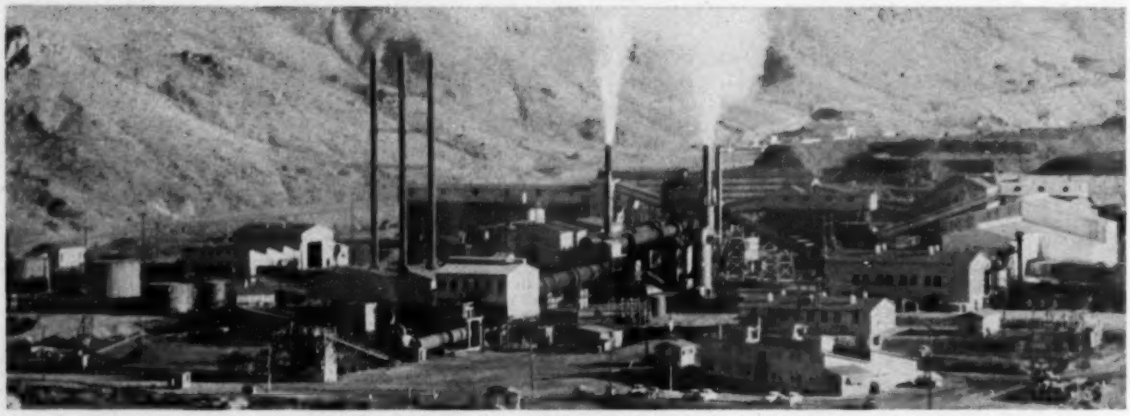

Fig. 2—Flotation mill and kiln plant, ore blending piles in background.

Experimental and operating results show the concentrate to lose from 12 to 15 pct by weight in the calcining and nodulizing process, owing to volatilization of CO_2, SO_2, and water of crystallization. An increase in manganese percentage is apparent and is depended upon in plant operation. No trouble is encountered when recoveries are calculated on a unit basis from the heads to the final nodule product over a period of time, because both products are weighed. The loss in weight does present problems in test work and analysis which will be discussed after the test procedures have been outlined.

The metallurgical data in Table I illustrate pilot plant results for one test. Upon nodulizing, the concentrate would show 15 pct insoluble, the upper limit on metallurgical grade nodules.

Table I. Pilot Plant Metallurgical Results, July 5, 1950

Product	Wt, Pct	Mn, Pct	SiO_2, Pct	Al_2O_3, Pct	Distribution
Dry ore	100.0	25.55			100
Rough conc't	64.9	36.36			
Cleaner conc't No. 1	58.1	39.08			
Cleaner conc't No. 2		39.59			
Cleaner conc't No. 3	50.2	42.30	10.67	2.64	83.1
Rougher tail	35.1	5.55			7.6
Cleaner tail No. 1	6.8	13.32			3.5
Cleaner tail No. 2	7.9	18.98			5.8
Cleaner tail No. 3	Recirculated				
		Ratio of Concentrate			1.98:1
Reagents Used		Lb per Ton			
Fuel oil		65.6			
Wetting agent		6.0			
Manganese sulphate (65 pct)		26.0			
Vegetable oil soap dry		47.6			
Sodium silicate		0.5			

Feed rate 1000 lb per hour. Lake Mead water used.
All reagents added separately to classifier overflow except 21.4 pct of total soap, which was added to No. 1 conditioner.

After completion of pilot plant flotation and nodulizing tests, work continued on reagent combinations and quantities in an attempt to make satisfactory recoveries on lower-grade ore and obtain a lower insoluble in the concentrate. Straight run and cracked oils were investigated as well as the effect of pulp temperatures.

An increase in the amount of fuel oil aided materially in lowering the insoluble. With the oil increased to 120 lb per ton, the dispersed condition of the concentrate disappeared. The concentrates assumed a granular form as the slimes coagulated, and the combined insoluble dropped to 8 to 10 pct. The oil emulsion method of manganese flotation has proved more stable and easier to control in plant operation than the dispersed method. Recent U. S. Bureau of Mines work[3] arrives at the same conclusions.

Soap skimmings, a byproduct of the sulphate paper industry, was not only the cheapest collector but gave equally good recovery and was available in the quantities required. The test work further indicated that an emulsion of oil, soap, and oronite-S added at the conditioning stage seemed to give better results than stage additions, so this procedure was adopted. Liquid SO_2 was added in a 3 pct water solution just ahead of the conditioning stage. The Ph value for conditioning and floating the ore is not critical and generally regulates itself between 7.8 and 8.6 pct, and total reagent consumption per ton of ore has averaged between 250 and 300 lb.

The elimination of reagents from samples to determine the exact manganese content has posed a problem, which as yet has not been solved in a satisfactory manner. Solvents do not easily remove the insoluble metallic soaps and oil by any sampling method rigid enough for test work or mill samples. The method now used is to dry the sample slowly, igniting the filter paper while the sample is still hot. The sample generally ignites itself and is partly calcined. The effect on the manganese content on laboratory samples is not serious, but wide variations did occur on the mill samples prepared in this manner. High assays occur, especially when unusually large amounts of reagents have been used, or when the ore being treated contains much of the sulphate minerals. Determination of intermediate mill products is therefore inaccurate; consequently the results of milling and nodulizing operations must be calculated on the basis of nodule units produced against crude ore units delivered to the mill.

Table II. High Oil Emulsion Laboratory Test No. 28, February 1952

Product	Wt, Pct	Mn, Pct	SiO_2, Pct	Al_2O_3, Pct	Pb, Pct	Distribution
Heads		21.31				
Redried concentrate	40.5	45.5	6.7	2.5		84.0
Combined middlings	8.4	12.1				4.6
Tails	39.9	4.7				8.4
Nodules	37.2	49.5	7.3	2.6	0.61	84.0
Reagents Used			Lb per Ton			
Dry St. Marys soap skimmings			40			
No. 1 fuel oil			120			
Oronite S wetting agent			6			
SO_2 as 5 pct sol.			8.5			

Conditioning 5 min, rough 3 min, clean four times. Conditioning at approximately 15 pct solids, flotation at 10 pct solids.

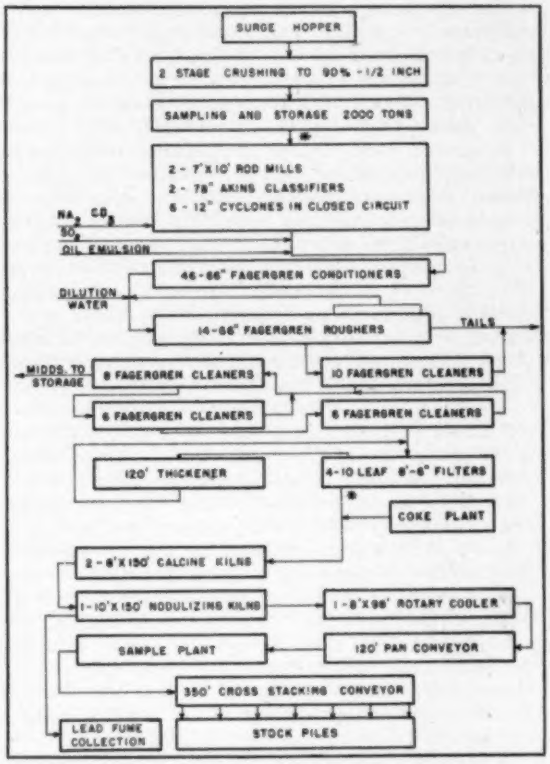

Fig. 3—Simplified flowsheet of major equipment and process steps at Three Kids mine.

A typical laboratory test using the high oil emulsion is shown in Table II. The reagents here amount to 166 lb per ton, and as nearly all this remains, reagents account for over 15 pct of final concentrate.

Conditioning in the laboratory is carried on in a Fagergren test cell for 5 to 10 min at densities of 15 to 30 pct solids. Although thousands of tests have been run in the laboratory at Manganese, Inc., the proper combination of power, motion, time, and air is still undetermined. The failure to correlate mill results with laboratory test work has led to many experimental changes in the mill conditioning. Even though improvements in conditioning have been made, the consumption of reagents in the mill is still greater than laboratory tests indicate they should be.

The plant is designed to treat 1200 long dry tons of ore containing 18 pct moisture. An area of more than 20 acres is required for the ore bins, crushing plant, flotation section, nodulizing kilns, conveyors, thickeners, shops, offices, and electrical substations. To conserve space a simplified flowsheet of the major equipment and process steps is shown in Fig. 3, and only those points which present unique problems will be discussed. Power is supplied from the lines of the Basic Management, Inc., at 69 kv to the main substation where the voltage is dropped to 2300 for distribution to three substations supplying the crushing plant, flotation plant, and kiln section. The primary crusher and rod mill motors operate on 2300 v, while the rest of the plant uses 440-v current.

Water is supplied from the 42-in. line of Basic Management, Inc., which crosses the property.

In the crushing section 5-in. discharge from the primary jaw crusher is scalped and the +½ in. is fed to one 30x30 double impact breaker where a product of 90 pct −½ in. is obtained in open circuit. Previous attempts to use cone-type crushers were unsuccessful, because of the pancake effect of squeezing the moist spongy ore. The tuffaceous ore is not too abrasive for the impact-type crusher.

Even though the ore contains 18 pct moisture when delivered from the mine, or about 12 pct from the stockpile, both crushing operations produce considerable dust. This problem is adequately taken care of by a Sly dust collector system.

Table III indicates the size of secondary crusher product. The percentage of fines has been higher than anticipated; therefore the plant now in construction employs ¼-in. rod deck screens to send +¼ in. directly to the mills and the undersize to the spiral classifier, from which the 10 mesh and finer material will overflow to cyclone classification, Fig. 4, where a 65 mesh separation is made. The +65 mesh returns to the mill for regrind.

Because of bentonite in the ore mechanical classification was impossible, even though the classifiers were lowered to a point where the weir and return discharge were at the same level and the pulp diluted to 12 pct solids. The introduction of cyclones in closed circuit with the mills and spirals has been the solution to a baffling problem.

Soda ash is fed in amounts of 5 to 20 lb per ton before the cyclones. Part of the gypsum dissolved in the grinding is thus precipitated and the tendency to form insoluble calcium soap after addition of the emulsion is reduced.

It is believed that when a relation is established between power, motion, dilution, and particle size the uncertainties of oxide manganese flotation will have been overcome.

Table III. Noncumulative Screen Sizing, Pct

Material	+¼	−¼	−10	−65	+65	+100	+200	−200
Impact crusher discharge	23	77	57	23				
Spiral class overflow					6.7	9.5	15.3	68.5
Cyclone overflow					0.4	2.1	7.5	90.0

On the basis of laboratory and pilot plant test work, 160 hp was the connected load for conditioning. The power and number of conditioners was gradually increased to 690 hp, at which time a maximum of 850 tons per day was treated. Sixteen additional cells using 480 hp were in the process of installation at the time of the fire. From past results

Fig. 4—The 12-in. cyclones shown above are used as classifiers in the grinding circuit.

Fig. 5—Interior of nodulizing kiln showing buildup of manganese on brickwork.

it is believed that the total of 1170 hp being installed in the new plant will give the conditioning necessary to bring the plant to full production.

Among other variables experimented with in the mill conditioners were corner and side baffles, underfeeding, low and high pulp levels, several types of impellers, and the introduction of small amounts of air and stage addition of reagents.

Fagergren rotors operating with a peripheral speed of 2250 fpm in 66-in. cells have been found to be the most effective conditioners. Perforated baffles placed at right angles to the sides of the tanks give the best results. The pulp level must be several inches over the rotors. Encouraging results have been obtained using stators around the rotors, with all air sealed off. Fourteen cells thus equipped are in use. The power used is only half that of the straight rotor cells, but the pulp level must be low and the retention time is reduced to a point where total efficiency is about the same as the rotor cells with double horsepower.

Conditioning time provided is 13 to 16 min, depending on exact pulp level and percent solids. Little difference in conditioning has been found in the range of 15 to 28 pct solids. Laboratory work has not indicated better conditioning with thicker pulp, difficult because of classifier limitations.

Effectiveness of conditioning is determined visually in the mill by dip samples taken with a beaker. If conditioning is complete the mineral will rise in a wide band and the gangue will settle on the bottom with a clear band of water between the two. Unless the band of water appears it is almost certain that the pulp is not conditioned enough for flotation.

Each flotation circuit uses seven 66-in. cells for roughing and four stages of cleaning. Some products are recirculated to supply water and stability to the circuit.

The conditioned pulp is diluted to 10 pct solids as it enters the roughers. Tailings are discharged at 5 to 6 pct solids. Dilution is essential in the cleaning. Feed and concentrate average 12 pct solids with tails showing 2.5, 1.7, 1.0, and 0.5 pct solids as cleaning progresses. A middling which amounts to 3.5 to 5 pct of the Mn in the ore is bled off and stored for future treatment.

With good conditioning the froth is a heavy matte about 2 in. thick and the bubbles have a bright appearance. An excess of reagents will flatten the froth to a scum, while a lack of reagents creates great volumes of froth without mineral flotation. The emulsion is fed by metering pumps with remote controls and rate indicators on the operating floor.

The Cattermole effect, so troublesome in Cuba,[4] has caused no difficulty in launders, although buildup is troublesome on the flotation cell stators. The oily concentrate causes all rubber and neoprene compounds to swell, and all such parts in contact with the pulp have been replaced with steel.

It has not been possible to operate a thickener on the mill concentrates. The repeated handling or flowing of the concentrate causes the heavy flocks to coagulate into beads or even balls, which stop up pumps and lines. Filters have been operated successfully by feeding the mill concentrate at 12 pct solids directly to the filter tanks. A certain amount of settling takes place and stirring of the tanks is necessary. When concentrate is fed directly to the filters there is a constant overflow of slimes which is thickened and returned to the filters.

Eight 12-in. cyclones have been installed above the filters to act as thickeners. The slime overflow is thickened in one 120-ft traction thickener, where a certain amount of coagulation takes place and then returns to the filters through the cyclones. Drags and classifiers similar to those used in Cuba[4] were considered for partial dewatering, but the variation in concentrate consistency from granular to dispersed indicated that poor separation would result. Cyclone tests indicate that even with highly dispersed pulp some thickening takes place. It is believed that the cyclone installation will increase the filter feed solids from 12 to 25 pct and thereby double the filtering capacity. Filter cake is voluminous and contains 30 to 35 pct moisture.

A small crushing and screening plant reduces stockpile petroleum coke to —⅜ in., which is fed in amounts of 3 to 5 pct of the dry concentrate weight to the filter cake conveyor.

The flow of wet filter cake and coke is diverted at 15-sec intervals between two 8x150-ft rotary kilns. Two pug mills were used to mix the coke and filter cake thoroughly. The discharge was fed through inclined pipes to the kilns. Stopping up of the feed pipes and backfeeding or spillage into the hood became so troublesome that ribbon-type screw conveyors are now being installed to do the job of mixing and feeding.

The wet filter cake is self-calcining after the kilns have been brought to 800°F by oil firing at the discharge end. Feed end temperatures range from 750° to 1200°F depending on cake moisture. Calcine discharges at 800° to 1000°F through alloy pipes to the nodulizing kiln.

As the hot calcine passes down the nodulizing kiln the temperature is raised to a maximum of 2400°F at a point 20 ft from the discharge end. From that point to the nose cold air chills the nodules at 2000°F or lower. About midway in the kiln the temperature reaches 1650°F and lead begins to volatilize. It is believed that the coke reduces the lead minerals to globules of metal which are then volatilized as one of the oxides. Simultaneously, part of the coke reacts with gypsum present to liberate SO_2, which in turn combines with the lead oxide fumes to form lead sulphate. This assumption is based on operating results. When a deficiency of coke or of air occurs in the kiln the lead content of the nodules immediately rises. When nodulizer feed is low in gypsum the lead in the fume occurs in nearly equal parts as oxide and sulphate. As the gypsum in the feed increases the proportion of lead as sulphate exceeds 90 pct and free SO_2 becomes a problem in the scrubbing system.

The combined insoluble also appears to have some control over lead liberation. When the insoluble approaches or exceeds 15 pct the lead is volatilized with difficulty. This effect may be due to the exclusion of air by the more fluid nature of the tumbling charge or to the chemical combination of lead compounds as silicates.

While 10 to 12 pct insoluble is desirable to make good nodules, more than that amount causes ringing and build-up in the kiln, Fig 5, which must be shot or bored out. On several occasions the kiln has been cooled down and the manganese coating removed with pavement breakers. Alternate raising and lowering of kiln temperature has been found effective in causing the coating to drop loose. The exothermic reactions of the coke midway in the nodulizer contribute to the ringing, as frequently the temperature at that point will be higher than at the expanded nodulizing zone near the firing end. Although other plants[4-6] have discovered that an expanded zone is of benefit as a preheater in nodulizing, it is found in this instance to be detrimental. The lack of control over temperatures at the kiln center results in pasty material sticking in the expanded section until the walls are the same diameter as the rest of the kiln. In other words, preheating begins with the exothermic reactions, and nodulization in this kiln begins near the center and continues for nearly 75 ft.

The hot and often sticky nodules drop through the firing hood to a rotary cooler. Air enters around the nodulizer nose and is drawn concurrently through the cooler by an induced draft fan. Spray water is introduced in the nodulizer-cooler hood as a cooling aid. Pan and belt conveyors carry the cooled nodules through a sampling plant to the nodule storage piles, shown in Fig. 6.

Dust from miscellaneous points around the kilns is collected and returned by air slide to the calcining kilns. The stack gases of each calcine kiln pass through a wet cyclonic scrubber and the resulting manganese slurry is recycled through the mill concentrate thickener and filter plant. This dust has averaged more than 100 tons per day, but the use of screw feeders on the calciners is expected to reduce the load appreciably.

Small amounts of lead fume precipitate in the nodulizer multiclone dust collector, but it has not been determined whether or not returned dust contributes to high lead in the nodules. From the nodulizer multiclone gases pass to wet cyclonic scrubbers, where the major part of the lead fumes is collected. A Pease Anthony venturi-type scrubber was very effective in precipitating the lead fume, but mechanical problems due to SO_2 corrosion and gas volumes larger than anticipated necessitated complete revision of the fume collection system.

Fig. 6—A view of storage piles and stacker boom.

The system now installed uses one 70,000-cfm hot fan and two refractory-lined wet cyclonic scrubbers of 12 ft 6 in. and 9 ft 6 in. diam in series. An increase in gas velocity through the multiclone by addition of fresh air is expected to remove most of the dust now lost in the lead sludge as indicated in Table IV.

Table IV. Extracts from Metallurgical Statement, 15 Days of June 1953

Product	Short Tons	Mn, Pct	SiO_2, Pct	Al_2O_3, Pct	Pb, Pct	Long Ton Units	Recovery, Pct
Heads	10,116	24.3	28.48	5.65	1.16	219,526	100.00
Middlings	1,076	11.2				10,802	4.94
Tails	4,360	9.3				36,372	16.46
Conc't	4,678	41.3	9.58	2.50	1.60	172,352	78.60
Nodules	3,609	48.3	11.71	2.43	0.67	161,362	73.50
Dust loss						10,990	5.10

Reagents	32 Plus Diesel Oil	63 Pct Soap	42 Pct Oronite S	SO_2	Na_2CO_3
Lb per ton	189.9	100.4	9.7	8.2	11.6
Total 15 days	1,921,359	1,015,225	98,221	83,360	117,600
Nodulizer fuel, Bunker C oil			363,827 lb	83.77 lb per ton	
Petroleum coke			382,894 lb	4.0890 of conc't	

Adequate storage facilities for reagents are on the property. Emulsion is prepared in a special plant, each ingredient being fed in the right quantity through metering pumps equipped with variable controls at a central panel. A 50/50 mixture of reagents with water is first made, diluted with water to a 20 pct strength, and pumped to the 50,000-gal mill storage tank.

Conclusion

Manganese oxide flotation is still very much an art. The flowsheet and operating techniques will continue to change. Oxide ores of 18 pct Mn can be treated commercially. It is believed that studies of the physical properties of the amorphous and microscopic crystalline oxides may indicate a solution to effective conditioning and thus lower the cutoff point on ores now amenable to flotation.

Acknowledgment

Since the project began, Hewitt S. West, President of Manganese, Inc., has directed the process development, which is based on the work of Robert Lord and variations developed by Frank Trotter and the Manganese, Inc., staff. Suggestions made by the staffs of the U. S. Bureau of Mines at Boulder City and Salt Lake City in developing the process contributed materially to its success. Identification of bentonite by Edwin De Moss, Mine Superintendent, and John Atkins, Staff Engineer, opened the way for improved classification.

The author wishes to thank F. A. McGonigle of Manganese, Inc., for review of the paper and permission to publish these data.

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Manufacture of Tungsten Carbide Tipped Drill Steel

by T. A. O'Hara

SINCE May 1948, when tungsten carbide bits were introduced at the Flin Flon mine, they have been popular with the miners because of their fast drilling speed and low gage loss. The high cost of commercial carbide bits and tipped drill steel, however, prevented their use except for the hardest rock.

In an effort to extend the use of tungsten carbide on a basis economically competitive with detachable steel bits, experimental work was begun in 1950 to test the feasibility of making tungsten carbide tipped drill steel in the mine drill steel shop. This work showed that tipped drill steel could be made locally at less than half the cost of the commercial product. The performance of the local tipped drill steel was comparable to that obtained with commercial carbide bits and tipped drill steel and the cost per foot drilled was much lower.

Local tipped drill steel was adopted for all mine drilling in November 1951. Since then drilling costs per foot have been sharply reduced and footage drilled per manshift has increased markedly.

Experience at Flin Flon has shown that production of satisfactory carbide tipped drill steel is not difficult and that highly skilled labor and costly equipment are not required. As long as wise selection of brazing materials is made and certain simple precautions are rigidly maintained, there is no reason why small mines with relatively unskilled labor cannot produce a satisfactory product.

The following description outlines the technique used at Flin Flon for making carbide tipped drill steel and discusses characteristics of the brazing process that make special precautions necessary.

Drill steel is forged to four-wing shape in a conventional steel sharpening forge. Standard steel dies are modified to minimize forging cracks around the central waterhole and to forge a blunt bithead on the steel. The steel is preheated to 1500°F and held at this temperature for at least 2 min. When the temperature has equalized throughout the steel section, the drill steel is transferred to the forging furnace and heated rapidly with a reducing flame up to 2000°F. This two-stage method of heating minimizes the grain growth and decarburization of the steel while ensuring that the steel temperature does not vary greatly throughout the forging zone.

After forging the steel is allowed to cool in air to about 1600°F before being annealed in a bath of vermiculite. Despite the high hardenability of the 3 pct Ni-Cr-Mo drill steel used, this simple treatment anneals the drill steel sufficiently for milling.

The forged and annealed drill steel is slotted on a plain horizontal milling machine that is equipped with a quick opening chuck and a slot depth stop. The full depth of the slot is milled in a single pass of the 3-in. milling cutter which is fed at 3¾ in. per min across the crown of each bit wing. The slots are cut to a width of 0.342 to 0.344 in. Maintenance of this slot width is necessary to ensure that the optimum brazing clearance of 0.002 in. will result after assembling of shims and carbide in the slot.

Prior to March 1953, when the milling machine was installed, drill steel was slotted on a small manually fed ¾ hp milling attachment mounted on the bed of a lathe. Over 16,000 drill steels were slotted on this unit, and in view of its small size and low cost it gave excellent service.

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Brazing of Tipped Steel

Drill steel that has been milled and cleaned in carbon tetrachloride is mounted in a rotating cradle holding six drill steels, the length of which may be from 2 to 12 ft. The slots in the drill steel, the shims, and the tungsten carbide inserts are thoroughly fluxed with a fluoride flux and assembled as shown in Fig. 1.

Fig. 2 shows the brazing equipment in use. As the ring burner is lowered over the bithead a spring valve opens the gas lines, and the gas mixture, preset to give a slightly reducing flame, is fed to the ring burner where it is lit from a pilot flame. The ring burner heats the drill steel over a zone about 1 to 2 in. below the bithead, which becomes heated by conduction through the steel.

By this means the bithead is heated rapidly and evenly, and contamination of the brazing joint with soot from the flame is avoided. The bithead is heated to the melting temperature of the brazing alloy within 1 min. This rapid heating minimizes the disadvantage of a non-eutectic brazing alloy.

The brazing alloy, a nickel-bearing quaternary alloy, is placed at the bottom of the slot below the carbide insert, as shown in Fig. 1. As the brazing alloy melts it is drawn by displacement by the carbide and by capillary action into all parts of the joint to displace liquid flux from metal surfaces.

As soon as the brazing alloy melts, each insert in turn is *wiped* by being moved back and forth along the slot. This action assists wetting of the carbide by the brazing alloy and assists in displacing molten flux from the joint.

After continuous heating for about 75 sec, when the bithead has reached a temperature of about 1500°F, the ring burner is raised and the gas supply is shut off automatically by the spring valve. As soon as heating is stopped a hand press is placed on the bithead and the inserts are squeezed down firmly. This action minimizes the clearance between the bottom of the insert and the slot. Correctly brazed steel should maintain a clearance at the bottom of the slot of 0.001 to 0.002 in.

After six steels have been brazed they are removed from the cradle and allowed to cool in air. As soon as each drill steel is cool it is dressed on a grinding wheel to remove excess flux and braze and is ground to the gage appropriate to the length of the drill steel.

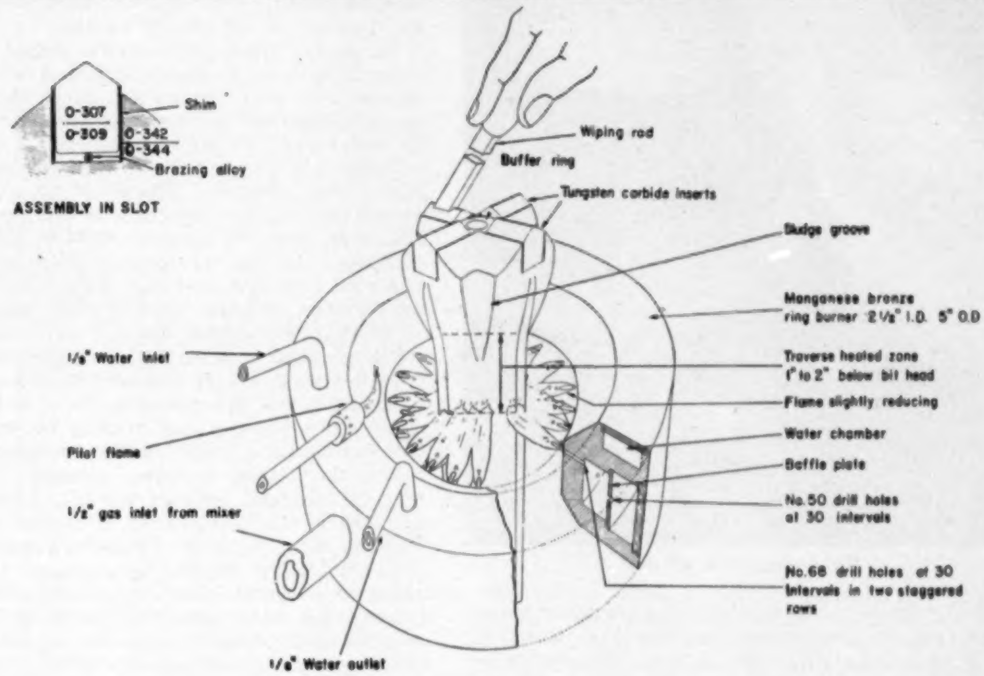


Fig. 1—The diagram above illustrates construction of the ring burner and assembly of the bithead.

The soundness of any brazed joint in a carbide bit, and consequently the performance of the bits in drilling, depends on the capacity of the brazing alloy to form a continuous solid layer between the steel and the carbide. Because of the difficulty of wetting tungsten carbide and the wide melting range of brazing alloys that wet carbide satisfactorily, a completely sound joint is rarely if ever obtained. Observations at Flin Flon on frequency of failures in commercial carbide bits and in locally tipped drill steel show that joint unsoundness is the primary cause of bit failure. Shattering or cracking of carbide tips is not necessarily caused by poor carbide, but frequently by failure of the supporting braze. Lack of soundness in a brazed joint may be brought about by any one of the following causes.

1—*Improper or Inadequate Cleaning of Carbide Inserts, Shims, or Drill Steel Slots.* All parts of the bit assembly should be thoroughly cleaned in carbon tetrachloride before assembly. At Flin Flon the brazing operator wears surgical rubber gloves when assembling inserts and shims in the bithead. This avoids contamination of the surfaces to be brazed with grease or sweat from the fingers.

2—*Improper Joint Clearance.* Long experience in industrial brazing shows that a joint clearance between 0.001 and 0.003 in. will give the soundest joint. In carbide-bit brazing, joint clearance at the sides of the slot is sometimes increased to accommodate shims for the relief of thermal stresses. Even where shims are used, the clearance between all surfaces to be brazed should not exceed 0.003 in.

3—*Improper Wetting of Carbide.* Joint unsoundness caused by improper wetting of tungsten carbide can be minimized by selection of suitable fluxes and brazing alloys and by correct technique during the brazing process.

4—*Slow Heating During Brazing.* Because all suitable brazing alloys are non-eutectic, rapid heating is necessary to minimize *liquation*. Liquation occurs when a brazing alloy separates into a low-melting fraction that is drawn into the joint, leaving behind a nickel-rich *skull* that may or may not become liquid at brazing temperature. The low melting fraction, containing little or no nickel, has poor carbide wetting ability, and joint unsoundness will therefore result.

5—*Too High a Brazing Temperature.* If quaternary alloys are heated to temperatures above 1500°F the zinc and cadmium in these alloys volatilizes, leaving gas pockets in the brazed joint, see Figs. 4a and 4b.

Thermal Stresses in the Brazed Joint

The rate of thermal expansion of tungsten carbide is about half that of steel; consequently the joint clearance of tungsten carbide inserts and the slot will increase on heating and decrease on cooling. During the austenite transformation of steel in the temperature range 1250° to 1350°F the steel shows an inversion in its thermal expansion, while the rate of thermal expansion of carbide remains constant. Fig. 3 shows the extent to which joint clearance will vary with temperature as long as carbide and steel are free to expand and contract.

It can be seen in Fig. 3 that when a tungsten carbide insert 5/16 in. wide is brazed in a steel slot at a temperature of 1200°F the amount of differential contraction on cooling will be 5/16x0.0053, or 1.006 in. A brazed layer of about 0.002 in., as required by conventional brazing practice, obviously cannot absorb more than a fraction of the differential contraction, and the remainder must be absorbed as compressive stresses in the steel and carbide adja-

Fig. 2—Brazing the drill steel.

cent to the joint. In a joint brazed without shims, these stresses may rupture the braze, Figs. 4c and 4d.

If a brazing alloy is used that solidifies within the critical range of the steel, the total differential contraction of the joint on cooling to room temperature will be lower, about 0.0012 in. for a 5/16-in. slot. However, the rate at which the tensile stresses accumulate during the critical range is much higher than the rate of accumulation of compressive stresses during cooling below the critical range. The resultant effect will be to increase the chance of premature failure of the brazed layer.

The effect of the thermal stresses can be minimized by three methods:

1—Use of a brazing alloy which solidifies below the critical range of the steel.

2—Slow cooling the brazed assembly to minimize localization of compressive stresses next to the joint.

3—Use of a joint artificially thickened by shims of ductile material which will absorb the compressive stresses without unduly increasing joint unsoundness.

Brazing With Shims

In the *sandwich* braze, copper or mild steel shims about 0.010 in. thick are placed between two pieces of brazing alloy foil and assembled on each side of the insert in the brazing assembly. Because an insufficient amount of brazing alloy cannot be added in foil form to fill the joint completely, *topping off* the joint with added brazing alloy is necessary as the brazing temperature is reached. Use of metal shims doubles the metal surface to be cleaned, fluxed, and brazed and increases the unsoundness of the joint. Tri-Clad materials, consisting of a copper or mild steel shim clad on both sides with brazing alloy, offer advantages over the sandwich braze. Fewer pieces are required in the bit assembly, and because the shim surface is already clad with brazing alloy, joint unsoundness is reduced.

Metallurgical examinations of the brazed bond and drilling tests have been made at Flin Flon on tipped drill steel brazed with plain steel shims, perforated steel shims, iron gauze, and tri-clad shims. These tests show that the least joint unsoundness and the best drilling performance result from using tri-clad shims that have been rolled with a corrugated surface to aid capillary action.

Successful brazing of carbide tipped drill steel depends as much on wise selection of brazing materials as on correct brazing technique. Moreover, the special properties of brazing materials required for carbide-bit brazing are such that only a small group of brazing alloys and fluxes can be used successfully. The properties desired of an alloy ideally suited for brazing carbide bits are good wetting ability on steel and tungsten carbide, narrow melting range, low flow temperature, high joint strength and good ductility, and high fluidity at the brazing temperature. No one brazing alloy, of course, has all these desired properties.

The problem of selecting a suitable brazing alloy is accentuated by the use of heat-treatable steels in the bit bodies of detachable bits or in tipped drill steel. This requires that brazing be conducted at temperatures and under cooling conditions that will induce the desired hardness, strength, and toughness in the steel body of the bit. Brazing alloys that can be used successfully in carbide-bit brazing will belong to one of the following groups.

1—The Nickel-Bearing Quaternary Alloys: This group of alloys is most widely used in carbide-bit brazing and is considered to possess the best compromise of the desired properties. An alloy typical of this group is comprised of 50 pct Ag, 15½ pct Cu, 15½ pct Zn, 16 pct Cd, and 3 pct Ni, with a melting point of 1195°F and a flow point of 1270°F. This alloy has good carbide and steel wetting properties, low melting temperature, good ductility and high joint strength, and fairly good fluidity at temperatures slightly above the flow temperature. Its wide melting range is a disadvantage that must be minimized by rapid heating. At temperatures above 1500°F zinc and cadmium in the alloy volatilize to form gas pockets in the brazed joint.

2—Nickel or Manganese-Bearing Ternary Alloys: An alloy typical of this group is comprised of 40 pct Ag, 30 pct Cu, 28 pct Zn, and 2 pct Ni, with a melting point of 1240°F and a flow point of 1435°F. Alloys of this group have good wetting properties for steel and carbide, good joint strength, good ductility, and fairly easy flowing characteristics, but higher application temperature and somewhat wider melting range make them less desirable than the nickel-bearing quaternary alloys for most carbide-bit brazing applications. Their higher melting temperatures may be useful where brazing temperatures of 1500° to 1600°F are necessary to induce the required hardness and strength in the steel bit body.

3—Nickel Brasses: These alloys have a melting range of 1700° to 1850°F. Analysis of a typical alloy is Cu 46½ pct, Zn 42½ pct, Ni 11 pct. These alloys have a strong wetting action on steel and carbide and form very strong joints, but these characteristics are offset by the high thermal stresses induced in the joint by cooling from the high brazing temperatures required. The sluggishness of these alloys throughout their melting range and the lack of ductility in the brazed joints formed restrict their usefulness for brazing carbide bits.

4—Copper: Copper has been used for brazing carbide tips in furnaces with reducing atmospheres, but its high melting temperature (1982°F) makes it unsuitable for torch or induction brazing.

Carbide tips are almost always brazed in air; consequently fluxes must be used to remove the oxides formed on the steel surface and to assist the wetting

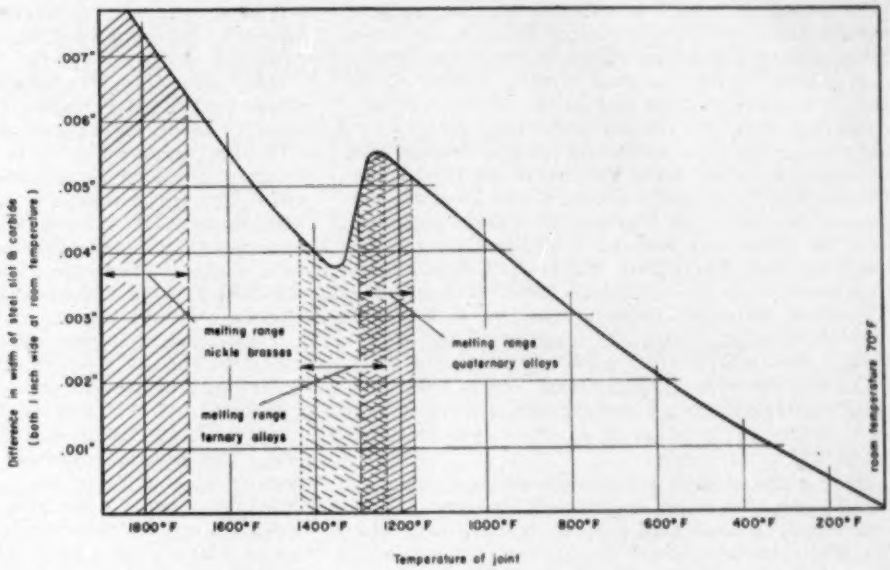


Fig. 3—The graph at left illustrates the differential contraction of steel and carbide.

of the metal surfaces by the brazing alloy. Although a flux with a low melting temperature is desirable to minimize oxidation of the metal surface, such low temperature fluxes are unstable at higher temperatures. In practice the most suitable fluxes melt at temperatures about 100°F below the melting point of the brazing alloy.

Fluxes

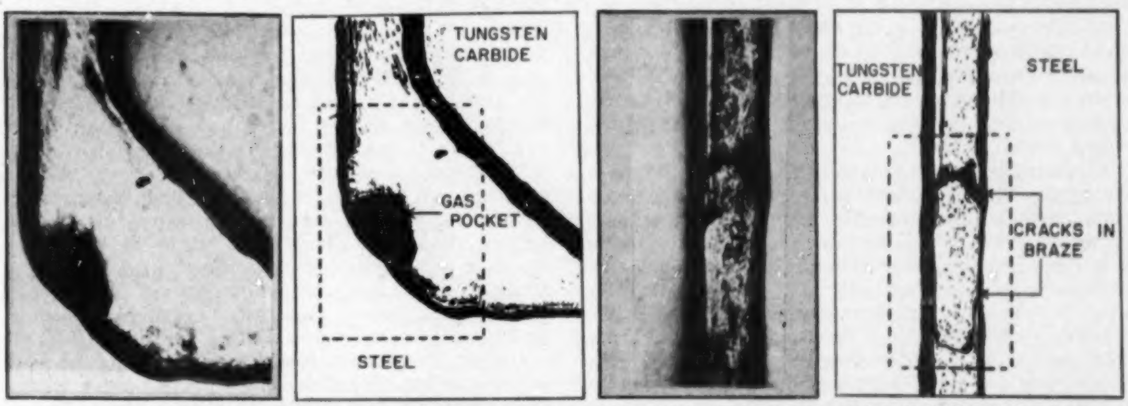
Fluoride fluxes are ideally suited for brazing with quaternary and ternary brazing alloys at temperatures less than 1500°F. At higher temperatures these fluxes are not stable and evolution of fluoride vapor causes undesirable working conditions. They are much stronger oxide solvents than the borax base fluxes. Fluoride fluxes containing sodium have an undesirable yellow glare when heated and fluxes containing potassium instead of sodium are to be preferred.

Borax-boric acid fluxes are very fluid when molten but are not strong oxide solvents, and they become sluggish when laden with oxides. These fluxes are useful in brazing with alloys which melt above 1500°F, but at lower temperatures the fluoride fluxes are much more suitable.

Tungsten carbide tipped drill steel must be cooled slowly from the brazing temperature if cracking of the carbide insert or braze failure is to be prevented. If cooling is too slow, however, the drill steel will not harden sufficiently to provide rigid support under the inserts. Tests at Flin Flon on drill steel cooled at different rates, resulting in Rockwell C hardness of from 24 to 53, show that satisfactory performance in drilling can be expected only when the hardness of the drill steel around the inserts is at least Rockwell C 40.

To obtain the high hardness required in the steel at a slow cooling rate, it is necessary to use an air-hardening drill steel. The drill steel used at Flin Flon is a medium carbon nickel alloy steel having a composition of 0.43 pct C, 0.60 pct Mn, 3.00 pct Ni, 0.40 pct Cr, and 0.25 pct Mo. This drill steel hardens satisfactorily by air cooling from 1500° to 1550°F to a Rockwell C hardness of about 50. The type of alloy steel formerly used at Flin Flon in shanks for detachable steel bits was a high carbon chrome alloy steel with the following composition: 0.95 pct C, 0.30 pct Mn, 1.00 pct Cr, and 0.25 pct Mo. Although the as-rolled hardness of this grade of steel is somewhat higher than the medium carbon nickel drill steel it does not air-harden as well.

Although the high carbon chrome type of drill steel is used extensively by Swedish tipped drill



Figs. 4a and 4b (left) illustrate a large gas pocket at corner of steel slot. Original magnification of Fig. 4a X100. Reduced approximately one-half. Not etched. Figs. 4c and 4d (right) illustrate rupture of braze resulting from thermal stresses in a bit brazed without shims. Original magnification of Fig. 4c X100. Reduced approximately one-half. Not etched.

steel manufacturers, it is believed that the medium carbon nickel drill steel is more suitable for torch brazing with quaternary silver brazing alloys.

It is well known that steel must be heated above the critical temperature and *soaked* at that temperature before it will harden completely on cooling. Soaking of the steel above the critical temperature is necessary because the steel must be completely transformed to austenite before it can be hardened on cooling. Alloying elements that form carbides, such as chromium and molybdenum, retard the austenite transformation. Thus alloy drill steels that derive their air-hardening power from alloyed chromium and molybdenum must be soaked for several minutes or heated to temperatures above 1600°F before they will air-harden satisfactorily.

On the other hand, alloy drill steels that air-harden chiefly because of their nickel content can be transformed to austenite at lower temperatures (1500°F).

Before tipped drill steel is placed in service at Flin Flon it is dressed, gaged, and hardened at the chuck end. Brazed drill steel is dressed on a C60 M5 VBE grinding wheel to remove excess braze, flux, and forging irregularities. Each insert is ground lightly on its outer edge to blunt the outer cutting edge. The outer periphery of the bithead is ground at 3° gage angle to the required gage on a 37 C60 K8 VK silicon-carbide grinding wheel. The required gage varies by 0.030 in. for each standard length of drill steel, being larger for the shorter lengths of drill steel. This ensures that new drill steel will not bind in a hole which has been drilled by a shorter worn drill steel. The standard gages for new drill steel are: 2 ft 6 in. steel, 1.50 in. gage; 4 ft 6 in. steel, 1.47 gage; 6 ft 6 in. steel, 1.44 in. gage; 8 ft 6 in. steel, 1.41 in. gage.

The chuck end of the drill steel is hardened to about Rockwell C 50 hardness by heating to 1475°F in pyrometer-controlled tube furnaces and cooling in air. The tipped drill steel is then placed in service. Tipped drill steel dulled in drilling is resharpened on a 37 C 60 K8 VK silicon-carbide grinding wheel.

Drill steel failures in service may be due to wearing out of carbide, loss of carbide, shattering of carbide, failure of steel around carbide, or breakage of the steel shank at the neck of the upset portion. The damaged end of these drill steels is cut off, serviceable inserts are salvaged, and the steel shank is reforged and re-tipped. Drill steel that has broken in the center of the shank away from the heat-affected zones on the bit and chuck ends and steel that has been shortened to 2 ft 0 in. long because of successive re-tipping is discarded. Eighty-eight per cent of all tipped drill steel made is used drill steel that may be re-tipped several times before being discarded.

The usual length of hole drilled at Flin Flon is 7 ft. Thus drifters with a 4-ft steel change can accommodate starters 3 ft 6 in. to 4 ft 6 in. long and followers 7 ft 6 in. to 8 ft 6 in. long. Stopers with 2-ft steel change require changes of 2 ft or less, and usable lengths of steel will include the following sizes: 2 ft 0 in. to 2 ft 6 in., 3 ft 9 in. to 4 ft 6 in., 5 ft 6 in. to 6 ft 6 in., and 7 ft 6 in. to 8 ft 6 in. Drill steel that is 1 ft shorter than the standard lengths of 2½, 4½, 6½, and 8½ ft is seldom useful in drilling, and this intermediate length will accumulate on the steel racks. Periodically this steel is cut down to the nearest standard size. Thus a new drill steel 8 ft 6 in. long will, on successive retippings, have lengths of 8 ft 3 in., 8 ft 0 in., 7 ft 9 in., and 6 ft 6 in., 6 ft 3 in.

Many mine operators believe that a drill steel becomes *fatigued* and useless for further drilling as soon as the inserts are worn out. Although there is very little support for this in practice or in theory, it serves to relieve the misgivings of mine operators when they accumulate a stock of used tipped drill steel. Some mine operators grind the bit end of this used steel to accommodate steel detachable bits, but the advantages of carbide drilling are lost. Retipping this used drill steel in the mine shops would be a practical way to utilize it. Most of it, unfortunately, is a high carbon chrome alloy drill steel, which is not as suitable as medium carbon nickel drill steel for tipping with a torch or ring burner.

All drill steel required for the Flin Flon and subsidiary mines is tipped in the Flin Flon drill steel shop. The average number of drill steel tipped monthly by a crew of five is 2037. The cost per tipped drill steel during 1952 was $6.40, which includes the cost of four carbide inserts. The average cost of four new inserts is $3.72, but carbide cost per drill steel is lowered somewhat by re-using salvaged inserts that are in good condition.

Drilling costs, including cost of tipping drill steel, sharpening, and nipping, averaged 7.7¢ per ft drilled during the first eight months of 1953. Table I shows the drilling costs and performance for the last six years. This table illustrates the increase in drilling performance since 1948, resulting chiefly from increased use of tungsten carbide bits or local tipped drill steel. Drilling costs increased when purchased carbide bits were used, but since 1952, when the local tipped drill steel was used for nearly all drilling, drilling costs have declined sharply.

Table I. Costs and Performance in Drilling at Flin Flon Mine

Year	Drilling with Tungsten Carbide, Pct	Footage Drilled per Driller Shift	Cost per Foot Drilled, ¢
1948	0.8	49.9	9.2
1949	18.5	51.5	9.9
1950	54.6 (Incl. 0.9 pct local)	61.5	11.6
1951	80.6 (Incl. 17.4 pct local)	69.2	11.0
1952	90.5 (Incl. 89.2 pct local	74.2	7.0
1953 (8 months)	99.8 (All local tipped)	83.5	7.7

Equipment necessary for carbide tipping drill steel need not be costly. Most mines have a forging furnace and a steel-sharpening forge, which can be utilized in forging the bithead. Few small mines have a milling machine, but unless a large amount of carbide tipped drill steel is required a small milling attachment should be adequate. Several small manually fed millers or milling attachments are available at a cost of $350 to $1200.

The conventional acetylene torch may be used for brazing carbide tips, although a ring burner will be much more satisfactory. The water-cooled ring burner shown in Fig. 1 was made in the mine machine shop and it is understood that somewhat similar burners are now available from suppliers of welding equipment. Induction heating is cleaner and faster than acetylene heating, but unless a large output of brazed drill steel is required the high cost of suitable motor generator equipment is not justified. Joints brazed carefully with a suitable gas burner need not be inferior to those brazed with induction equipment.

Trends in Coal Utilization and Their Effect On Coal Marketing

by Carroll F. Hardy

The day by day loss of industrial plants to gas and oil is chiefly by default. The coal industry is not selling its superior economy, safety, and other advantages to its customers.

THE position of the coal industry has been affected by a wide variety of developments in the production and use of energy. The tempo of development and change has been increasing and the end is not in sight. Legislation is currently being proposed for commercial use of atomic power, and the employment of atomic energy in significant quantity will probably occur about the same time as the decline in production of petroleum and natural gas. But these developments are in the future and have little immediate effect on utilization and marketing of coal. While no one should try to suppress or retard the development of a new and economical source of energy, both the coal and private utility industry should be allowed to question how the nuclear power is to be used, who is to use it, and who is going to pay for it. The taxpayers have a monopoly on fissionable material and the knowledge to employ it. Any commercial use must stem from this source. It is not hard to visualize either taxpayer-subsidized private utility atomic power plants on one hand and super TVA's on the other.

In view of the gains of gas and oil in the home heating field, it is interesting to compare the 1940 and 1950 census reports on the kind of fuel used for heating in occupied dwelling units. Table I shows that whereas coal provided 77 pct of the fuel for central heating (furnaces and boilers) in 1940, it was down to 45.4 pct in 1950. However, only about 1½ million units were lost in this 10-year period.

In the non-central heating category, which principally includes stoves, the percentage declined from 39.2 to 25.6, but the units declined about 2½ million in number. The big increase was in heating units designed to burn gas and oil. Use of wood for central heating declined about one-third.

Data on amount of fuels used for residential heating are not available, but information is on hand for residential and commercial space heating, see Table II. Commercial space heating includes office buildings, churches, schools, and similar structures. The annual use of bituminous coal in these two categories declined about 1 million tons in the 10-year period. Other forms of solid fuel showed greater losses, except wood, which remained the same.

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Domestic stokers reached their high point in 1948 with about 1,200,000 in use. At the end of 1951 there were approximately 1,116,790 stokers in use. Conversions to gas and oil have been from hand-fired heating plants in the ratio of about 7 to 1 compared to stokers. In other words, for every one stoker which has been converted to gas or oil, seven hand-fired units have been converted to gas or oil. A bare recital of these data would indicate that the coal industry is holding its own reasonably well. However, 93.4 pct of the new homes built in 1951 were heated by gas or oil. Oil-burning equipment was installed in 37.8 pct and gas equipment in 55.6 pct of the new homes. This indicates that the public prefers gas when it is available, and that oil is second choice, with all forms of solid fuel apparently used when it is unavoidable.

It must be pointed out, however, that during the period of rapid expansion of gas pipelines gas has been sold for house heating at prices that are in some cases actually lower than coal prices, or very nearly on a par. Gas has been sold at wells at far below the comparable price for oil produced from the same wells, and far below its actual worth. This situation is being remedied at the present time by increases in gas prices at the wells. For example, the wellhead price of gas in Texas averaged 7.49¢ per Mcf in 1952. In 1949 it was 4.59¢ per Mcf. This increase in price is being reflected in pipeline gas prices, and in most of the markets served by the pipelines the tendency is to get it out of the *bargain basement* type of sales.

The American Gas Association estimates that at the end of 1952 there were in the United States about 11 million customers for gas house-heating, and the Association expects additional gains each year until around 18 million homes will be heated by gas in 1975. By 1975 there should be 60 million dwelling units to be heated in the United States, if dwelling units increase at the same rate as the population. If the gas industry heats 18 million dwelling units by that time, this still leaves 42 million units to be heated by some other fuel. If oil is used to heat 18 million dwelling units in that same year, 24 million would of necessity be heated by coal, coke, wood, electricity, or another fuel. The total number of dwelling units using coal listed in the 1950 Census was 18,776,000, so it would appear that coal has a chance at least to stand still in the tonnage sold for domestic use.

In the first quarter of 1953, 2044 domestic stokers

were sold. This equals 25 pct of the sales in 1949. To recover the former sales volume the coal industry must actively promote and sell the present units and also develop and promote units more acceptable to the public.

Trends in Industrial and Commercial Utilization

There are two principal trends in the general industrial and commercial fields. One of these is the trend away from hand-firing and the handling of coal by shovel and wheelbarrow, and the other is away from coal. Only a few industrial plants are still hand-firing coal, and these in general are in seasonal industries such as canning factories, or smaller and older plants. It was long felt that in as much as a fireman had to be on duty in any case, he might as well hand-fire the coal and move coal and ashes by wheelbarrow. However, because of the stoker's superior efficiency, this is not a valid assumption. It is also true that younger men, particularly, do not care to work in a hand-fired plant. They prefer a less arduous job.

Table I. Heating Fuel for Occupied Dwelling Units in the United States Urban and Rural for Years 1950 and 1940

Central Heating	1950	Pct	1940	Pct
Number reporting fuel	20,724,000	100	14,152,024	100
Coal	9,430,000	45.5	10,903,163	77
Wood	250,000	1.2	373,322	2.6
Gas	5,914,000	28.5	1,100,587	7.8
Utility gas	5,719,000	27.6		
Bottle gas	195,000	0.9		
Liquid and other fuels	5,130,000	24.8	1,765,952	12.5
Liquid fuel	4,678,000	22.6		
Other fuel	452,000	2.2		
Non-Central Heating	**1950**	**Pct**	**1940**	**Pct**
Number reporting fuel	20,053,000	100	19,469,707	100
Coal	5,127,000	25.6	7,622,427	39.2
Wood	3,969,000	19.8	7,362,155	37.8
Gas	5,881,000	29.3	2,728,381	14.0
Utility gas	5,204,000	26.0		
Bottle gas	676,000	3.4		
Liquid and other fuels	5,077,000	25.3	1,756,740	9.0
Liquid fuel	4,437,000	22.1		
Electricity	272,000	1.4		
Other fuel	369,000	1.8		

The same general thought applies to stoker-fired plants where coal must be wheeled to the boiler room and then shoveled into the stoker hopper. Coal-handling equipment has proved its value, even in those plants where there is a minimum labor force. The fireman has a chance to concentrate on keeping the stoker running at top efficiency, and should, in theory at least, cut down on the waste of fuel by inefficient operation.

In the smaller plants, automatic ash-handling equipment cannot be justified because of the relatively few ashes to be handled. In the medium-sized and larger plants, some types of ash-handling equipment usually can be justified.

The trend away from coal towards gas and oil is caused by the same factors behind the trend toward gas and oil in domestic heating. Increases in mine and freight costs, strikes, labor costs, smoke abatement activity, trouble in the use of coal due to improper selection, poor coal deliveries—all are reasons for dissatisfaction. A dissatisfied user is a poor prospect. In many plants management has been plagued by undersized boilers, improper design, worn-out equipment, and a whole parade of incompetent firemen, as well as by direct troubles with coal. A *packaged boiler* which has complete control and may be started and stopped by push button offers quite a contrast. The boiler is delivered to the plant with the burner and controls already in it. It is skidded into place, the oil or gas line and the steam line are attached, the stack connection is made, and it is ready to go. Management is often told that no supervision is necessary, that there will be little or no maintenance, and that once such an installation is made their problems are all solved. They are assured that aside from an occasional check as an extra precaution, the boiler room will be self-operating.

The packaged boiler has had a very wide acceptance, in fact, a much wider acceptance than an examination of all the factors involved would warrant. It is not maintenance-free, nor is it foolproof. It is true that the packaged boiler designed for gas or oil is lower in first cost than a standard boiler and stoker combination complete with automatic controls and automatic coal-handling equipment. However, it is often overlooked that a saving of 12 pct, for example, in investment cost will be wiped out in a year or less by the higher cost of the fuels used. In most of the industrialized part of the United States, coal is by far the lowest cost fuel.

In those areas where gas is available on an interruptible basis, it can sometimes be shown to be economical to burn. However, there must be a standby fuel in this case. Oil is used for standby in packaged boilers, on the assumption that the gas will be off for only a short period of each year and that the high cost of oil, because of the small quantity used, will be of no consequence.

The price of interruptible gas is usually tied to coal, and gas is sold at about the same price or perhaps a little less than the cost of coal. Oil, except in a narrow area on the eastern seaboard, is usually one and a half to two times as expensive as coal. Therefore, when gas is not available and oil must be used, apparent savings of using gas are quickly overcome. In cases where gas is competitive, coal can be used as a standby at somewhat higher cost installation than a straight packaged boiler, and full economy is effected. It has been true for many years that the amount of gas available on an interruptible basis dwindles as the pipeline is loaded up. To design a boiler which may be in use for 30 years or more to use interruptible gas with oil standby is sheer folly, whether it is a packaged boiler or a standard boiler.

There are many cases on record where packaged boilers have been put in and then taken out when they proved uneconomical, with a loss to the customer which could have been prevented had the full facts been known. Standard boilers installed to use gas or oil and not designed for easy conversion to coal present a serious problem also. It is possible to use coal in a modern fully equipped installation with no more labor force than is required in a combination boiler burning gas and oil. Until this fact is firmly established in the minds of consulting engineers, architects, and fuel users, the industry will continue to lose business in this group. The whole coal industry must expend more effort in *education* and promotion to stop or even slow down the trend away from coal in the industrial field.

The effect of the partial dieselization of all but one of the country's major railroads and of most of the smaller railroads is well known. It has eliminated much of the run-of-mine, resultant, and double-screened tonnage that was sold for locomotive fuel. The trend towards eliminating the coal-

fired steam locomotive has certainly not stopped and may not stop until some of the major railroads are completely dieselized.

In addition to providing a market for about one-fifth the total bituminous coal produced, consumption of railroad fuel had another important function. The practice of allowing the producer to ship almost any size of coal for locomotive fuel was an accommodation to the producer and allowed him to get rid of no-bills and canceled cars. This balance wheel, generally speaking, has been lost, in addition to the loss of 120 million tons of railroad fuel.

Table II. Fuel Consumption for All Residential Uses and for Commercial Space Heating, According to Physical State at Point of Combustion for Years 1940 and 1950

Liquid Fuels (Thousands of Barrels)	1940	Pct	1950	Pct
Distillate grades 1, 2, 3, and 4	115,533	10.1	220,947	14.7
Range oil No. 1, fuel oil, and kerosene	44,692	3.8	94,662	6.2
Residual grades 5 and 6	44,846	4.3	72,716	5.3
Total	205,071	18.2	388,325	26.2
Gaseous fuels (millions of therms)				
Natural gas				
Residential	4,064.3	6.1	11,561.6	13.2
Commercial	1,226.2	2.7	3,496.6	4.0
Manufacturing gas including mixed gas				
Residential	1,758.8	2.7	2,277.5	2.6
Commercial	371.5	0.6	607.2	0.7
Liquefied petroleum gas	128.0	0.2	1,931.5	2.2
Total	7,548.8	11.4	19,874.4	22.7
Solid fuels (wood, thousands of cords; all others thousands of net tons)				
Bituminous and lignite	87,700	34.6	86,604	26.0
Anthracite	39,300	15.1	29,558	8.6
Coke	8,231	3.2	2,560	0.8
Briquets and packaged fuels	1,313	0.5	2,730	0.7
Petroleum coke	844	0.4	1,201	0.4
Wood, including mill waste	48,500	15.3	48,739	11.7
Total		69.1		48.2
Electricity (thousands of kilowatt hours)				
Produced from fuel	16,865,569	0.9	52,733,944	2.1
Hydro	8,442,908	0.4	21,696,464	0.8
Total	25,308,477	1.3	74,430,408	2.9

The rapid expansion of electric utilities is an old story although an almost unbelievable one. The expansion of electric utilities in the eastern half of the United States will require between 30 and 40 million tons of additional coal by 1956, if all the units planned are built. This is a very encouraging trend, but not a completely happy condition.

Over a 25-year period the electric utilities have reduced rates very materially and have maintained these low rates even in the face of rising costs of labor, fuel, and other materials. Part of this has been done by the improvement in fuel-burning equipment. This is shown by the amount of coal required to produce 1 kw-hr of electricity. In 1930, for example, it took 1.6 lb of coal per kw-hr; in 1940, 1.34 lb per kw-hr; in 1950, 1.19 lb per kw-hr; in 1952, 1.10 lb per kw-hr, and in March 1953, 1.08 lb per kw-hr. The effect of this reduction in coal rate from the 1.19 lb average in 1950 to the 1.08 lb average in March 1953 is just over 10 pct. That is, if fuel required per kilowatt-hour in March 1953 had been at the 1950 average rate, 1,600,000 more tons (or equivalent) would have been used. On the basis of a full year at this rate, over 19 million tons more coal or coal equivalent in gas and oil would be required. This reduction in fuel requirements per kilowatt-hour has been common to all three fuels and has been accomplished by higher steam pressures and temperatures, better cooling of electric generators, better combustion controls.

The most modern steam electric plant has a heat conversion efficiency of 37 pct. In other words, it uses 37 pct of the heat in the fuel. Announcement was recently made by the American Gas and Electric Co. of a boiler designed to operate at 4500 psi steam pressure and 1150°F. This boiler will be much more efficient than the present boilers, although the exact efficiency is not revealed.

The trend in designing furnaces has been towards more liberal or flexible design, so that a wide variety of coals may be burned, more specifically, so that the lower-grade coals may be burned and the pressures still be maintained on the boiler. This trend, which started in the public utilities, has gradually spread to large and small industrial plants. Thus as the coal industry has spent $2 billion in the past 10 years in underground and surface production facilities and cleaning plants, to give a better and more uniform product, the coal customers have spent an equal or greater amount for equipment which would use a lower-grade fuel. Part of this trend for wider flexibility has stemmed from the desire to use locally produced coals and thus save the freight rate on better-grade coals brought in from a distance.

Large industrial plants have followed the lead of the utilities in boiler design and coal-buying practices. Both classes of plants have had troubles such as slagging of tube banks, clinkering, and coal and ash handling difficulties, from using poor coal. Large utility plants have been put out of service by low-grade coals. In less extreme cases extra maintenance and lower operating capacity have cost more than the savings on the delivered cost of the low-grade coal versus good coal.

The Trend of the Delivered Price of Coal

Table III is a compilation of the coals bought by a large midwestern plant using approximately 280,000 tons of coal per year. The plant was new and modern in 1937 and the equipment is essentially the same today. The same type of coal has always been used in this plant, as coal selection is somewhat limited by furnace design.

There are several interesting points about this table. Column *A* gives the yearly average Btu per pound as bought. In 1937 the average was 13,575; in 1938 it had risen to 13,790. The low point was in 1949 when the average had dropped to 12,865. In 1952 it was 13,175. There have been many rumors about the decline and quality of better-grade coals because good seams are all mined out or because mechanical mining has supplanted hand loading. Actually, those who spread the rumors know that this is not the case. High-quality coals are as available today as they were before World War II. There have been some changes in quality of individual mines, and some mines in good seams have mined out as have mines in poor seams. In the case of a particular plant, after the demise of fixed prices and controls on coal, higher Btu coals than the ones used were available. However, it was the decision of management to use the somewhat lower cost and lower Btu coals.

Column *B* of Table III shows the average mine price paid during the 16-year period. It will be noted that the mine price has declined exactly $1.00 per ton between 1948 and 1952.

Column *D* shows that the average delivered price has declined only 32¢ in that same period, although owing to increases in freight rates the delivered price increased in 1949 over 1948.

Column C shows an increase in freight over the 15-year period since 1937 of $1.55 a ton. The present freight rate is $3.66, so by the end of 1952 the freight had increased 75¢ a ton over the rate in 1948. In this same four-year period, the cost of labor at the mines had increased $1.05 a ton. The average delivered cost per ton in 1952 was 32¢ less than it was in 1948 and 48¢ less than it was in 1949. The producers have received nothing for their increase in labor cost since 1948 and have in effect absorbed the increase in freight so that they are receiving the equivalent of $2.12 less per ton than they were in 1948. The profit or margin in 1948 was only a small part of this $2.12, so that the producer has had to cut production cost in many ways even to stay in business.

As C. J. Potter recently pointed out, with the tonnage off about 28 pct from the peak year of 1947, and with prospects of further decreases, many coal operations have had to go out of business. Stronger elements, as mentioned above, have made major investments in mining and preparation facilities and major reductions in labor costs. The financially weaker elements of the industry have turned to opportunists' methods of mining, or have attempted to operate non-union, or have closed.

With production off 70 million tons since 1951 and prospects for further decline in tonnage for 1953, together with the fact that the coal industry's profits are approaching or below the zero point, a revolution of coal mining or marketing must, of necessity, be in the making.

The Effect on Coal Marketing

A purchasing agent said recently, "Coal marketing consists of trying to cut the price and go broke quicker than the other coal companies can do it."

Table III. Cost and Heating Value of Coal Used by a Large Midwestern Plant, 1937 to 1952 Inclusive

Year	Avg Btu Per Lb	Avg Mine Price, $	Avg* Freight, $	Avg Deliv. Cost Per Ton, $	Avg Cost Per MM Btu, ¢
	A	B	C	D	E
1937	13575	1.69	2.04	3.73	13.75
1938	13790	1.64	2.14	3.78	13.70
1939	13650	1.59	2.14	3.73	13.65
1940	13750	1.61	2.14	3.85	14.00
1941	13635	2.17	2.14	4.31	15.75
1942	13615	2.34	2.23	4.57	16.80
1943	13470	2.70	2.23	4.93	18.30
1944	13275	3.01	2.23	5.24	19.75
1945	13225	3.04	2.23	5.29	20.00
1946	13185	3.30	2.37	5.67	21.50
1947	12985	4.19	2.56	6.82	26.25
1948	12915	5.23	2.91	8.14	31.50
1949	12865	5.08	3.22	8.30	32.25
1950	13200	4.56	3.27	7.83	29.65
1951	13165	4.56	3.36	7.92	30.08
1952	13175	4.23	3.59*	7.82	29.89

* All increases in freight rates were applied as they became effective and the costs averaged for the year. The 4¢ transportation tax is included.

Coal salesmen are continually offering to cut prices to move coal, or are ready to meet any cost per million Btu figure regardless of the actual use value of the coal. Purchasing agents are naturally either skeptical about the fact that the coal industry is not making money or think they might as well get in on a good thing as long as coal is being sold at a loss. The fact that coal cannot continue to be sold at less than the cost of production does not worry them, since they are regularly offered more tonnage than they can use, at lower and lower prices. This inept marketing on the part of the coal salesmen has encouraged many buyers arbitrarily to set a cost per million Btu and hold all suppliers to this price regardless of coal quality. They are trying to beat down the prices which are already too low. These customers of the coal industry evidently do not realize that much of the reduced cost of mining coal in the last few years has seriously harmed mine operations. It has produced no capital for replacement or renewal and has paid nothing for the money used. The present over-capacity of the coal industry has permitted this disorderly marketing —if it may be called marketing—to exist, and has permitted the buyer to write his own ticket.

Fortunately, not all purchasing agents and their managements feel this way about coal. They have given some consideration to the future supply of coal as well as to the present situation. In a recent comprehensive statement on this subject, Robert E. Dennis, Fuel Agent of the Consolidated Edison Co., said in part:

> So let us recognize that coal is our basic, most dependable source of energy for static power plants. And let us see to it that the coal industry is not so injured by loss of business to liquid and gas fuels as to make it impossible for it to absorb a large and rapid demand to supply the energy needs of our basic industries, which employ static power plants.
>
> As the needs for more power come upon us in the future, it's a cinch that coal will have to supply the bulk of it. What we have to do now and perhaps next year, is to help the coal industry over this period. After that, it is my opinion coal *can* be able to meet all competition, due to the demand for coal from the utility and chemical industries. To keep the coal industry healthy and competitive, it needs, above all else, tonnage sufficient to meet its inherent mining load factory. . . . In the first and most important place, there is need for recognition on the part of all industry that, in harming coal they are jeopardizing the future security of this country, and of course, themselves. Of what use are millions upon millions of tons of coal near at hand, with no one willing to put up the money, no one willing to put in the labor to obtain it? I am the buyer for a company whose needs for fuel are enormous, and whose struggles for more economical ways to give better service are unceasing. Nevertheless, we see clearly that today's economy may be tomorrow's disaster and, we must, therefore, have strong, efficient industries to supply and transport our fuels and, in so far as we can do so, help them to stay that way.

The coal industry is faced with the task of educating all consumers of coal towards full realization of the true conditions of the industry and the consequences of irresponsible buying and selling of coal.

The Task of Marketing Coal Properly

The consumer of coal must be brought to realize that a large proportion of the producing capacity of the industry is made up of thoroughly reliable companies which can be depended upon in any emergency. This might be three-fourths, or more, or less, of the industry. These are the companies that have an investment in the industry in the form of adequate reserves, adequate mining, and preparation machinery. These are the people who make every effort to mine coal safely, prepare it properly, and market it consistently.

The rest of the industry is made up of opportunists who saw a chance to get into coal production during the war and when new tonnages opened up, for example, the T. V. A. tonnage. Their only interest is to try to make a quick dollar by excavating and selling coal without preparing or standardizing.

In this day of exact purchasing, of careful testing, of micrometer measuring, the method of coal buying in most companies is badly out of step with the rest of the material procurement. The steel used by a company is usually bought on specification from a large steel company. It is then tested and analyzed to make sure that it is up to specification before it is put in the product. Every effort is made to purchase from reliable, established companies. The jobber with a few tons of steel or a few barrels of unbranded paint has little chance of selling to a careful buyer. The buyer would certainly want the analysis, the origin of the material, and probably the reason it was on the market at reduced prices.

Table IV. Comparative Annual Fuel Costs in a Plant Requiring 695.5 Billion Btu Annually

Coal Analysis, As-Received	COAL A 1¼x0 In.		COAL B 1¼x0 In.
Moisture, pct	3.95		6.18
Ash, pct	3.97		9.55
Sulphur, pct	0.70		4.49
Ash softening temp., °F	2700		2070
Btu per lb	13910		11740
Operating Costs			
Tons of coal required	25,000		29,620
Mine price of coal, $	4.70		2.90
Freight rate, $	4.70		4.70
Deliv. cost per ton, $	9.40		7.60
Deliv. cost per million btu, $	0.3379		0.3237
Total deliv. fuel cost, $	235,000.00		225,112.00
Storage loss, weathering, fires, etc. (3 M tons stg.) 5 pct, $	141.00	15¢ per ton	3,420.00
Unloading and handling labor, 30¢ per ton, $	7,500.00	30¢ per ton	8,886.00
Coal handling equip. maintenance, 5¢ per ton, $	1,250.00	15¢ per ton	4,443.00
Soot blowing and slag removal, 1¢ per ton, $	250.00	5¢ per ton	1,481.00
Pressure parts and fly-ash collection erosion, 2¢ per ton, $	500.00	8¢ per ton	2,369.60
Corrosion of fly-ash collector, fans and stack, $		Per year	2,000.00
Ash-handling equip. operation and maintenance, 1¢ per ton, $	250.00	5¢ per ton	1,481.00
Fly-ash return system maintenance, 1¢ per ton, $	250.00	6¢ per ton	1,777.20
Grate and feeder maintenance, 3¢ per ton, $	750.00	14¢ per ton	4,146.80
Total operating cost, $	10,891.00		30,004.60
Total fuel and operating cost, $	245,891.00		255,116.60
Utilization cost per ton, $	9.84		8.61
Utilization cost per million btu, ¢	0.3537		0.3670
Annual Savings by Using Coal A, $	9,225.60		

Coal buying, unfortunately, is a different proposition with many of these same buyers. They appear to care little about the reliability of the company or even the source and analysis of the coal, and they evince small concern as to the probable result of using coal in the plant, or the extra cost that will ensue when a number of different coals are mixed together. It is not at all unusual to find a plant using coals varying in ash content from 3 to 12 pct, in moisture from 4 to 15 pct, and in Btu content from more than 14,000 per lb to 11,000 per lb in a random mixture as the coals are received at the plant. Even the best of operating personnel with the best equipment is sorely taxed to operate a plant properly under these conditions. There is a trend towards avoiding any long-term relationship with well-established coal companies, but dependence on day-to-day purchases from anyone with a few cars of coal to sell.

A recent study of a moderate-sized plant with flexible coal-burning equipment is shown in Table IV. This illustrates the difference between the cost of coal f. o. b. the plant and the complete cost of using that coal removing the ashes. The costs given in this table are based on actual experience and all estimates shown are conservative. Even in other cases where the low-ash, low-sulphur, high-Btu coal might be shown to be slightly more expensive, the human relations or good labor policy should be well worth this small cost.

Formerly it was the practice to build a factory for only one purpose—to get out production. No attention was given to the architecture of the plant or the appearance of its surroundings. If it happened to be an eyesore, as most factories were in those days, this was merely considered part of the price of having it in that location. Today this idea has given way to the esthetic or *good neighbor* policy. Factories are carefully designed and grounds landscaped. Usually both improve the neighborhood.

The purchase of good standard coal from reliable coal companies is only another step in this program. A large part of the air pollution can be prevented in this manner, and certainly the advantages well offset any slight addition in cost.

Summary

The trend in coal utilization is definitely towards use of the smaller sizes of coal. It is probable that as time goes on this will be even more pronounced. To stay healthy, the coal industry must realize enough money from the sale of these sizes to make a profit. Profit is the only known reason for private capital being invested in any operation. The coal industry's customers must be made to realize that a healthy coal industry is not only an asset but a necessity to all American industry. The alternative is wholesale liquidation of the coal industry which would be disastrous to the economy of the country.

The coal industry cannot hope to cut costs by mechanization, selective mining, augers, continuous miners, or stripping as fast as the realization can be cut by poor marketing and incorrect application.

Losses of residential and commercial heating tonnage to gas and oil are serious, although new equipment may regain part of this loss in time. The coal industry must develop, test, promote, sell, and service such equipment. This can only be done through united effort.

Widespread use of the coal-fired combustion gas turbine locomotive will recover a part of the railroad fuel business, when this locomotive is put into production.

Promotional work, including better engineering of coal-burning equipment, is a very important part of the marketing of coal. The work of spreading the gospel of proper design of coal-burning plants, proper maintenance and supervision, and careful consideration of coal in all plans for expansion and in new plants must be doubled and redoubled. Coal salesmen must be experts in coal selling and application—not expert salesmen who can sell appliances, hairpins, or real estate with equal facility.

There must be full recognition of the present situation of the coal industry, of the trends which affect marketing and the courses open to change these trends. Responsible coal producers and sales companies must make a concerted effort to convince their responsible customers that a fair and adequate price for coal is a good investment. It cannot be done by hoping that someone else will do it.

Frontiers in Heat Extraction from the Combustion Gases of Coal

by Elmer R. Kaiser

COMBUSTION of coal and transfer of heat from flames and gases to boiler surfaces continue to be of great interest to engineers here and abroad. Numerous investigations have been in progress to improve furnace and boiler performance and economy. The importance of better understanding of the processes and opportunities for improvement is apparent when it is remembered that heat from at least 500 million tons of coal a year the world over is being transferred to boiler water at efficiencies ranging mostly between 50 and 90 pct. Even slight gains in efficiency, economy, and labor saving become very significant when multiplied by the enormous quantity of fuel consumed. Also the competitive position of the large coal, oil, and gas industries in satisfying the fuel consumers is greatly affected by the achievements made through technical progress with each fuel.

This paper is part of a continuing activity of Bituminous Coal Research, Inc., to extend the knowledge of coal utilization for steam generation and to seek promising directions for future research and development in cooperation with others. Particularly in the latter regard, numerous interviews were held during the last three years to seek the experience and advice of boiler and combustion-equipment manufacturers, electric-utility executives, and fuel engineers. A wealth of published information was also reviewed, which together with the interviews pointed to the advisability of further work on ash and sulphur control.

For the present purpose a number of factors important to efficient heat liberation and recovery have been grouped as follows: 1—combustion, temperatures, and rates of heat liberation; 2—radiation, convection, and furnace and boiler configuration; 3—sponge ash, slag, and hard-bonded deposits; 4—low-temperature deposits and corrosion (cooling flue gas below dew point and air-pollution control); 5—the limitations of coal cleaning and boiler size and cost as related to fuel characteristics; 6—future possibilities and conclusions.

The development of combustion apparatus for power boilers is progressing at a lively pace. There has been no letup in improvements in design of pulverized-coal-fired boilers, and there is a strong trend at present toward improving dry-bottom units. Spreader stokers with overfire jets and dust collectors as standard equipment are gaining favor. Less than 10 years in commercial use, cyclone burners are going into numerous installations here[1] and abroad.[2] Underfeed and traveling-grate stokers have long since been developed for heavy-duty operation, yet new developments in overfire jets and humidification of air blast have improved their performance. A water-cooled vibrating-grate stoker of German origin is being introduced into the United States and Canada.[3]

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The primary objectives of an ideal coal combustion device are: capacity to burn the variety and sizes of coals likely to be economically available during the life of the unit; capacity to burn the coals automatically for a wide load range and rapid load fluctuations and to burn the coals completely to CO_2, H_2O, and SO_2, which means without smoke and cinders, or carbon in the refuse; capacity to control and discharge all the ash in final granular form without ash adhesion to walls or tubes, and without flue dust; minimum furnace volume; minimum labor and maintenance; low initial and operating cost.

Regardless of the method of burning, the gaseous products of coal combustion are N_2, CO_2, O_2, H_2O, and SO_2. By way of illustration, the coal analyses in Table I is assumed from an installation described by E. McCarthy.[4]

Table I. Products of Coal Combustion

Proximate Analysis, Pct		Ultimate Analysis, Pct	
Moisture	8.0	Moisture	8.0
Volatile matter	35.0	Carbon	67.5
Fixed carbon	47.0	Hydrogen	4.6
Ash	10.0	Nitrogen	1.4
		Oxygen	5.2
Total	100.0	Sulphur	3.3
		Ash	10.0
Btu per lb: 12,360		Total	100.0

When coal is burned with 20 pct excess air (theoretical air, 9.23 lb per lb of coal), the quantities of combustion gas shown in Table II are produced. In addition, the gases carry particles of fly ash, unconsumed cinders, soot particles, and small but significant amounts of vaporized oxides and sulphates of sodium, potassium, lithium, phosphorous, iron, and other metals. In recent years, germanium, one of the rare metals found in coal, has been shown to oxidize and vaporize at combustion temperatures and to be concentrated by recondensation at lower temperatures.[5]

Pulverized coal and cyclone flames[6] have peak temperatures of 3000° to 3500°F. Temperatures in fuel beds of large underfeed stokers reach maxima of 3000°F, sufficient to fuse almost any ash and to volatilize some of it. These peak temperatures are above the optimum necessary for rapid combustion, but they hasten heat transfer for ignition as well as boiler heat absorption. Furnace and gas temperatures increase with combustion air preheat. Low excess air has the same effect. Fine coal pulverization and highly turbulent combustion shorten the distance for fuel burnout, increase flame temperature, and speed up heat transfer.

Rates of combustion of pulverized coal exceeding 200,000 Btu per cu ft per hr have been demonstrated in atmospheric gas-turbine combusters,

vortex burners, and primary furnaces of boilers. The heat release rate in the cyclone furnace exceeds 400,000 Btu per cu ft per hr.[6] Similar heat releases can be obtained in fuel beds. With each of these types of combustion units it has been necessary to provide additional furnace volume to achieve 95 pct or higher completion of combustion. The hourly heat release per cubic feet of total furnace volume frequently ranges between 15,000 and 21,000 Btu for pulverized-coal-fired boilers.[4] Above 27,000 to 28,000 Btu per cu ft per hr the carbon loss may become excessive. Combustion rate is purposely low to insure adequate cooling of gases and fly ash by heat transfer to the furnace walls before the gases enter the convection banks.

With good mixing of fuel and air and adequate time for combustion, modern power boilers have reduced losses of unburned fuel of 1 pct or less, and excess air is only 20 to 25 pct.

Radiation, Convection, Furnace and Boiler Configuration

An excellent and recent review of the factors and mathematics of heat transfer in furnaces of water-tube boilers has been prepared by G. G. Thurlow.[7] The Research Committee on Furnace Performance Factors sponsored by the American Society of Mechanical Engineers has investigated not only the total heat absorption but the area heat-transfer rates in the furnace walls.[8-11]

Approximately half the heat liberated in a boiler is absorbed by the water-tubed furnace surfaces. Boiler furnaces do not operate under black-body conditions; the heat absorbed by radiation in a pulverized-coal-fired boiler is 0.6 or less that which would be absorbed under black-body conditions at the gas exit. The emissivity of particles, dust clouds, flames, and boiler surfaces has been studied by many investigators. Currently a cooperative program with international sponsorship is in progress in Holland to study flame radiation.[12]

Table II. Quantities of Combustion Gas Produced by Burning Coal With 20 Pct Air

Gas	Dry Volume, Pct	Lb Per Lb Coal	Lb Per Ton Coal	Volume of Gas at 330°F, Cu Ft Per Lb Coal
N_2	81.2	8.537	17,074	181.5
CO_2	15.0	2.475	4,950	33.5
O_2	3.6	0.428	856	8.0
H_2O		0.494	988	16.3
SO_2*	0.2	0.066	132	0.6
	100.0	12.000	24,000	239.9

* If all S were burned to SO_2, the dry volume percentage would be 0.27. Actually, some SO_3 and sulphates are formed.

In a modern pulverized-coal-fired boiler furnace with metal wall surfaces, heat is transferred from the flames to tube surfaces which are at temperatures up to 1200°F. Heat transfer is principally by radiation. Convection heat transfer is kept low, 3 to 14 pct, to reduce the rate of ash deposition on tubes and to prevent overheating of cleaned areas.

The design problem then becomes one of proportioning the furnace and locating the burners in the furnace walls to complete combustion, prevent direct impingement of gases and fly ash on the walls, and transfer enough heat to the water and steam to cool the products of combustion at least 150°F, and preferably 200°F, below the ash-softening temperature before the gases enter convection tube banks. It is well known that slag screens, divided furnaces, and tall furnaces accomplish these objectives. Also, installation of the convection tubes beyond a right-angle turn of the furnace exit gases prevents direct flame radiation on the tubes. This helps to prevent slag deposits on convection tubing.

A newer development coming into use is the tube curtain or platen, a partition of tangent water tubes extending into the furnace space to receive heat from both sides. Figs. 1 and 2 illustrate one application of tube curtains to cool furnace exit gases.[13] A number of boiler manufacturers apply platens for waterwalls, for convection surface, and for superheaters.

In an application, not illustrated, at the new Oak Creek Station of the Wisconsin Electric Power Co., three partition walls along the front wall form four semi-cells between down burners. Not only will the partition walls absorb heat from the pulverized-coal flames while combustion progresses, but the superheat and reheat temperatures can be controlled by differential firing into the semi-cells.[14] The boiler was designed for coal with a 2000°F ash-softening temperature.

The most critical condition is in the superheater and reheater tubing, which in some cases contains steam flowing at 1050°F. With steam inside the tube the heat transfer rate of 75,000 Btu an hr per sq ft of wall surface is an allowable maximum (300°F gradient through tube wall), while 40,000 Btu an hr per sq ft (150°F gradient through tube wall) is a desirable rate. Present practice for such boilers is 45,000 to 50,000 Btu per sq ft per hr, and future practice is predicted to be 35,000 to 40,000 Btu per sq ft per hr. Boilers with waterwalls may have heat absorption rates as high as 75,000 Btu per sq ft per hr.

Conventional boilers for high pressure operation use 3-in. diam tubes with thick walls which are more easily overstressed by heating than are tubes of smaller diameter and thinner walls. The development of boilers with pump-stimulated circulation of water has made practical the use of thinner tubes of smaller diameter.

The heat absorbed in the superheating and reheating sections of many units is 40 pct or more of the total heat input. With steam temperatures rising an average of 12°F a year, the trend is said to be inescapably toward more superheating and reheating in the furnace-wall tubes. The importance of radiant superheaters to supplement and balance convection superheaters is well known. Not all manufacturers agree on a trend toward radiant wall superheat and reheat.

Assuming clean tube surfaces at 1200°F, a heat transfer rate of 40,000 Btu per sq ft per hr by radiation only, and a combined radiation coefficient of 0.75, it is possible to calculate a mean temperature for the furnace by employing the Stephan-Bolzmann equation:[15]

$$q = k\left[\left(\frac{T_1}{1000}\right)^4 - \left(\frac{T_2}{1000}\right)^4\right]$$

where $q = 40{,}000$; $k = 1720 \times 0.75 = 1290$, a constant for the furnace conditions; and $T_2 = 1200 + 460 = 1660°\ R$, outside tube temperature, then solving for $T_1 = 2490°R$ or 2030°F, which is the mean furnace temperature.

In other terms, the mean temperature of the surfaces and flame *seen* by a unit area of wall under consideration must be only about 2030°F if the heat transfer rate is to be 40,000 Btu per sq ft per hr. The center of the flame will be much hotter, but other walls seen by the unit area will be only 1200°F or less, with a combined equivalent of 2030°F.

$$\begin{aligned} \text{When } q &= 50{,}000,\ T_1 = 2140°\text{F} \\ q &= 60{,}000,\ T_1 = 2250°\text{F} \\ q &= 70{,}000,\ T_1 = 2350°\text{F} \\ q &= 80{,}000,\ T_1 = 2430°\text{F} \\ q &= 90{,}000,\ T_1 = 2510°\text{F} \\ q &= 100{,}000,\ T_1 = 2580°\text{F} \end{aligned}$$

The present trend in heat absorption rates in dry-bottom furnaces is said to be downward in the electric-utility industry.

The lower furnace temperatures required in future practice will be obtained by lower heat absorption rates per square foot of furnace wall. Heat must be radiated from the burning particles and absorbed by the walls as the combustion of the particles proceeds. To illustrate the trend, design data are given in Table III for two plants of the Wisconsin Electric Power Co.[15]

Table III. Design Data for Heating Units, Wisconsin Electric Co.

Stations	Port Washington	Oak Creek
Year	1945	1953
Steam temperature, °F	900	1,000
Btu per sq ft per hr of furnace surface	85,000	65,000
Radiant heat absorption by superheater and reheater, Btu per sq ft per hr	42,000	32,000
Ash-softening temperature, °F	2,100	2,000

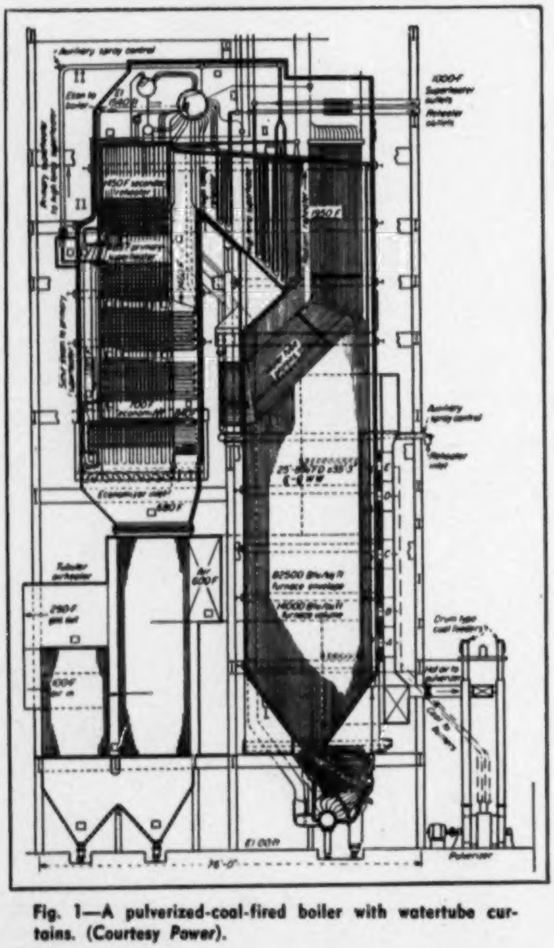

Fig. 1—A pulverized-coal-fired boiler with watertube curtains. (Courtesy *Power*).

Control of furnace temperature is also being achieved in some modern boilers by recirculation of flue gas and by adjustable burners for selectively using a part or all of the available furnace volume.

Information collected from several sources indicates a definite trend toward lower gas temperatures leaving the furnace to prevent spongelike deposits and slag on hotter tubes. Lower heat release and absorption rates are projected to assist in ash control and to prevent overheating of furnace tubes carrying high-temperature steam.

Sponge Ash, Slag, and Hard-Bonded Deposits

Sponge Ash and Slag: For most coal-fired boilers there is no serious problem with tight ash deposits on heat transfer surfaces. Where high ratings are carried and where ash-softening temperatures are low, the problem is more critical. Outstanding advances have been made in boiler equipment to prevent or control tight ash deposits.

Ash deposits on boiler surfaces insulate heat and in convection passes restrict gas flow. Since the outer surface of the slag is considerably hotter than the boiler tubes, heat flow is still further reduced. Reduced heat transfer causes higher flame temperature which in turn increases slag formation.

When coal is burned in pulverized form, all the ash is initially in suspension in the furnace gases, 50 to 70 pct or more of the ash being carried into the convection passes. With industrial stoker firing the quantity of solids suspended in the gases after combustion weigh between 5 and 40 pct of the weight of ash in the coal fired.[16] Only a minor fraction of dust leaving the furnaces with the gases is deposited on the boiler tubes, but the dust can form loose or tight coatings depending on temperatures and dust compositions.

Fig. 3 illustrates a cyclone-fired boiler, unique with respect to ash in that only 10 to 15 pct of the ash is carried out of the cyclone in suspension with the gases.[17]

The deposition of the dust on boiler tubes has been studied in detail by numerous investigators. The work done for the British Boiler Availability Committee is outstanding among the more recent studies of ash on boiler surfaces. The outside surface of the tubes is a thin film of oxide of the tube metal. The next layer on the tubes is a thin film of sulphates of iron, sodium, and potassium. These water-soluble compounds of low softening temperature form a base for the adhesion of particles of fly ash that reach the furnace tubes in dry or molten form. Molten ash particles sometimes impinge, fuse, and solidify on the tube metal or surface oxide.

The particles on the tube surface have a higher temperature than the tubes. When additional particles of dust deposit in successive layers, the temperatures rise and the particles gradually sinter together to form a surface that is both plastic and sticky. Surface tension and chemical action between the particles promote sintering and the evolution of gas respectively, with the result that a spongy mass is formed. The effect is most evident on the furnace walls and entrance to the convection zone. Elsewhere the temperatures are seldom high enough for vitrification, the deposit being dry and loose.

Delayed burnout of the coarser particles of pulverized coal, long flames, and near-flame impingement on wall surfaces increases rate of deposition because the resultant ash particles are hotter and stickier when they strike the tubes.

Ash particles vary in composition and softening temperature. Analyses of deposits of fly ash and sponge ash show average compositions of SiO_2, Al_2O_3, CaO, Fe_2O_3, and alkalis. A more detailed study of the compositions and fusion characteristics of the individual particles of mineral matter would probably reveal the way some act as cementing agents for others.

Once the buildup of ash deposits has begun, rate of deposition is accelerated. Deposits reduce the rate of heat absorption; hence gas temperature increases. Increased temperature promotes ash deposition and adherence and extends the zone of sintered deposits to surfaces farther along in the path of the gases.

Unless such deposits can be broken off by decrepitation, as by dropping load, or by lances and jets, the boiler must be taken off the line for cooling and manual cleaning.

Studies have been made of the behavior of ash after the layer has become thick and hot enough to flow. The viscosity of molten ash has been measured by P. Nicholls and W. T. Reid.[18] Messrs. Reid and P. Cohen reported on studies by the Bureau of Mines on factors affecting the thickness and flow characteristics of ash on furnace wall tubes.[19]

Mumford and Bice[20] have estimated that heat transfer to furnace tubes varies with the ash-deposit thickness as shown in Table IV.

Table IV. Slag-Ash Factors

Avg Thickness of Slag or Ash Deposit, In.	Factor
0 to 1/32	1.0
1/32 to ¼	0.7
¼ to 1	0.5
1 and over	0.3

Measurements made in a spreader-stoker-fired boiler, Fig. 4, show that dirty waterwalls can reduce heat absorption by 20 pct for the entire furnace.[6] In pulverized-coal firing, DeLorenzi states the furnace exit gases will be 200°F, or more, lower when the furnace walls are clean than when the walls are dirty.[21]

Clean tube surfaces show a period of immunity from deposits, sometimes up to six weeks, but in rare cases not at all. Tubes have been coated with aluminum, graphite, and lime with some beneficial effect for the life of the coating.

Boiler-tube corrosion from deposits is a serious problem and considerable progress has been made in alleviating it in the furnace. Information developed on corrosion of tubes will assist in an understanding of the nature of deposits next to the metal. A difference of opinion exists on whether the alkali sulphides cause tube wastage in reducing zones. Deposits of iron sulphide have caused trouble.

Ash may be classified into three categories: silicates, sulphides, and vaporized alkalis. The objective is to burn the sulphides and retain all the ash ingredients in the form of silicates. FeS and other sulphides are not miscible with slag.

It has been reported[22] that silicon sulphide (SiS) forms readily in a stoker fuel bed by reaction of

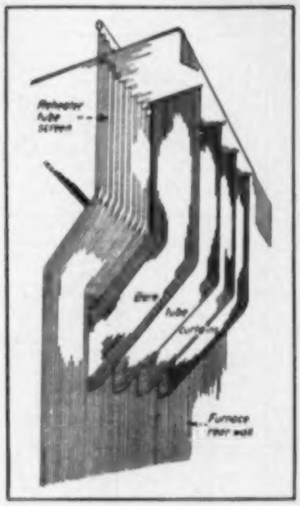

Fig. 2—Curtains of bare tubes in tangent contact. (Courtesy *Power*)

FeS_2 and SiO_2, especially if the oxygen content is low. The SiS sublimes at 1400°F and may deposit on surfaces below 1400°F. SiS is sticky at temperatures prevailing on superheater tubes and may cause ash particles to adhere. Oxygen and steam decompose it. Whether the same effect occurs in pulverized-coal firing is doubtful, but it should not be overlooked as a possibility.

The methods of removing slag from the furnace walls and the entrance to the convection passes are well known. Dropping the load temporarily has a marked effect. Soot blowers and wall deslaggers using either steam or air have been highly developed for the purpose. In troublesome zones not readily reached by soot blowers hand lances are employed.

By trial and error, users of coals from various sources determine which coals have ash characteristics suitable to their plants and loads. Unpredictable clinkering and slagging sometimes occur when mixtures of coals from different sources are consumed. Blowing loose ash off boiler tubes is not regarded as a problem. In general, users learn to adapt to the ash problem in older high-duty boilers, and select new boilers with more conservative furnace conditions. Nevertheless, it may be advisable to re-examine the limited knowledge of ash and develop 1—ash slagging tests for coals and 2—techniques for preventing adherent ash deposits.

Hard Bonded Deposits: The ash deposits most difficult to remove are the so-called bonded or alkali-matrix deposits that form on economizer, preheater, or superheater, and steaming-bank tubes. These deposits occur where the temperature of the flue gases is below the softening temperature of the ash. The fact that hard, bonded, enamel-like deposits that cannot be removed effectively by soot blowers can form on surfaces from 150° to 1100°F gives some idea of the wide scope of the problem. In cases where deposits form in superheaters, economizers, and air heaters, the condition of the scale indicates that chemical attack of the metal surface and of the ash itself takes place. Practical operating techniques to prevent these deposits are apparently not known, other than to increase excess air and to change coals. Fortunately the difficulty is limited to relatively few boilers and coals.

Chemical analyses of the deposits indicate the chief ingredient to be a sodium-potassium pyrosul-

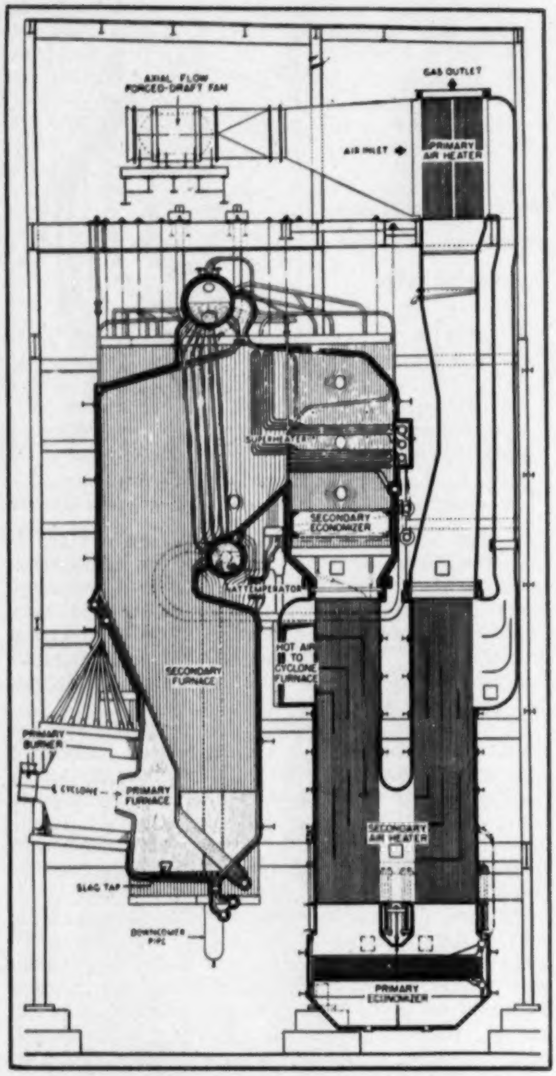

Fig. 3—A cyclone-furnace steam generator. (Courtesy *Power Generation*).

phate. The sulphate is probably the result of chemical reaction in the gas space of SO_3 with the alkali metals sublimed in the heat of the furnace. The sulphate then condenses on the tube where it forms a sticky surface to trap other particles of ash. The sulphate has a lower melting temperature than the silicates comprising the bulk of the ash.

The deposits are more severe in stoker-fired and cyclone-fired boilers than in pulverized-coal-fired boilers. The higher ratio of fly ash in the latter probably dilutes the sulphate deposit and acts, according to E. G. Bailey, like "flour on a pie board" in preventing bonding. The dry ash particles are also believed to form suspended nuclei for the condensation of alkali sulphates, rendering the sulphates harmless and carrying them into the dust collectors in dry form. Nevertheless, hard-bonded sulphate deposits occur in some pulverized-coal-fired boilers. Some plants in the United States have reported deposits containing as much as 40 pct sulphur trioxide even though the coal contained less than 1 pct sulphur.[illegible]

In the cooler zones of the steam generator, such as primary superheater and especially air preheater, the sulphate deposits are also accompanied by corrosion of the metal. High dew points of flue gas because of SO_3 are believed to be responsible for both corrosion and deposits. The sulphuric acid attacks the metal and traps and holds dust particles. Alumina is attacked to form aluminum sulphate, which is sticky and collects more fly ash or dust.

Sulphate-type deposits have been giving increasing troubles in the past few years and may be partly the result of the higher efficiencies and lower excess-air requirements of the modern installations. In a study by Barkley, Burdick, and Berk of the Bureau of Mines,[illegible] the deposits decreased as the CO_2 decreased. With 9 pct CO_2, only about one-third as much deposit was laid down as when 15 pct CO_2 was carried at about the same dew point. A boiler with economizer and air heater shows very little change in efficiency with minor changes in excess air. Enough excess air should be used to avoid troublesome deposits. Normal boiler design is based on 13.5 to 15 pct CO_2, varying with the coal.

Manual cleaning and water washing are employed to remove the deposits. The deposits weaken by water soaking, but much physical effort is required before the surfaces are clean. Regarding both sponge ash and bonded deposits, the British Boiler Availability Committee[illegible] recommended 1—that boiler surfaces be thoroughly cleaned of ash down to bare metal and 2—that ash buildup should not be permitted to increase draft loss by over 50 pct.

A specially designed pellet gun has been developed recently for removing slag deposits in very high-duty furnaces and in furnaces using particularly low-quality fuel.[illegible] The gun is said to be effective against deposits originating in a cyclone furnace fired by a coal containing chlorides. Asphalt-asbestos pellets, ½-in. diam, are propelled by compressed air at rates of 50 to 250 pellets per sec to remove unusual deposits.

In Europe the Broman system of raining hard iron pellets over boiler tubing has been adapted to 90 boilers.[illegible] The pellets are recovered and re-used. Other developments involving tube vibration, ultrasonic agglomeration, and nozzle contour are currently being investigated.[illegible]

The ultimate solution to the problem of the bonded deposits will probably be found in chemistry. Much work remains to be done to determine the causes and cures. Sponge ash can be controlled better through boiler design than can the bonded deposit. A program of research of no small magnitude is indicated, including also the conditions of boiler operation that promote and prevent the deposits. The fact that pulverized-coal-fired boilers are relatively freer of these deposits than are stoker- and cyclone-fired boilers suggests that refractory dusts of fly ash might be added to the furnace atmospheres.

At one large British power station phosphatic deposits on heating surfaces, due to vaporization of phosphoric acid from the fuel bed, were overcome by introduction of pulverized coal above the stokers and recirculating part of the flue gas.[illegible]

Low-Temperature Deposits and Corrosion

Modern power boilers of high efficiency recover heat from boiler flue gases, first by means of economizers which preheat feedwater and then by preheaters of combustion air. The lowest practical

limit of cooling today is set by the temperature of the coldest preheater metal in contact with flue gas, i.e., where the cold inlet air enters the air heater. The metal temperature should be safely above the dew point of the flue gases.

At and below the dew point, sulphuric acid of high strength condenses and rapidly corrodes preheater steel. At the same time the acid forms sticky areas on which dust deposits build, ultimately clogging the gas passages. The sulphate of the preheater metal forms a deposit which contributes to the clogging and reduction of heat transfer. Because of the soluble nature of the deposits, periodic washing with hot water is effective.

The sulphuric acid (H_2SO_4) is produced when SO_3, usually under 0.02 pct, combines with water vapor in the flue gas. Even when only 1 or 2 pct of the sulphur in the coal burns to sulphur trioxide, the dew point is raised significantly. The dew point is not affected by SO_2 and only to a minor degree by the amount of water vapor normally present,[29] as shown by Fig. 5.

The high concentration of the sulphuric acid condensate is illustrated by reference to Fig. 6 from Johnstone.[30] For example, condensate produced between temperatures 300° and 150°F would have the approximate compositions between points A and B.

An electrical dew point meter has been developed,[31] but dust from the gases affects the dew point film and renders accurate determinations difficult.[32]

Many researchers have investigated the chemical mechanism and kinetics of SO_3 formation. The high temperatures in the furnace are unfavorable to SO_3 production, but between the furnace and preheater the iron oxide on boiler tubes may catalyze the reaction $2SO_2 + O_2 \rightarrow 2SO_3$ (Harlow). Ingredients in fly ash and the secondary combustion of small amounts of carbon monoxide (Whittingham) may also promote SO_3 formation. Gumz[33] has recently

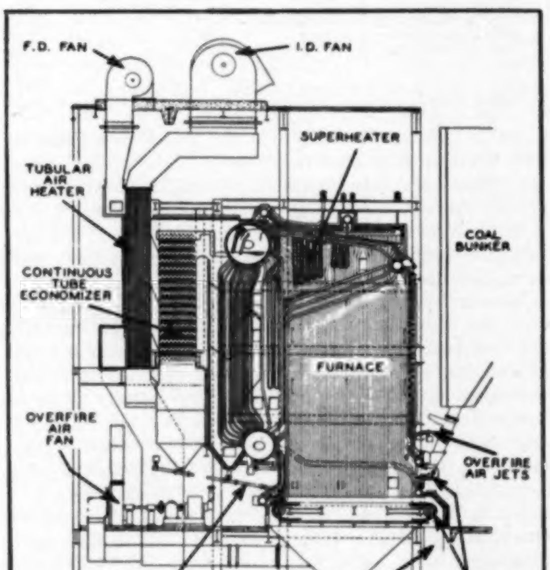

Fig. 4—A spreader-stoker-fired test boiler. (Courtesy American Society of Mechanical Engineers).

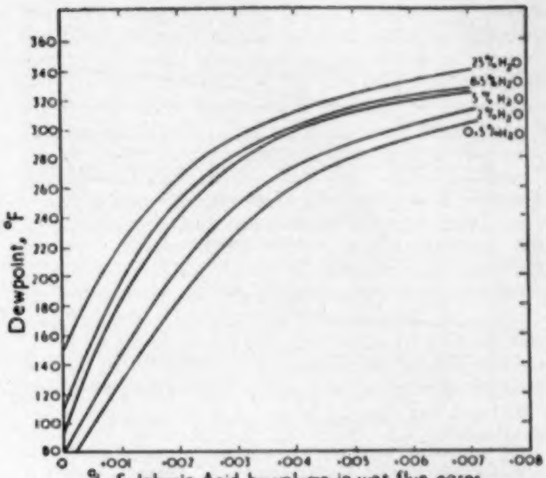

Fig. 5—Variation of dew point with H_2SO_4 content for gases having different water-vapor contents. (Courtesy British Coal Utilisation Research Association).

summarized the equilibrium data and pertinent references. Much more remains to be done to explain the chemical processes and to develop suitable controls. It is believed that corrosion and deposits can be lessened by reducing the sulphur in the fuel and by preventing the formation of SO_3."

"To investigate and report on the problems of corrosion and fouling in equipment exposed to low-temperature flue gas" is the scope of the ASME Research Committee on Low Temperature Flue Gas Corrosion and Deposits, which was organized in 1951 with John F. Barkley as chairman.

A recent Bureau of Mines report[34] on corrosion and deposits in regenerative air preheaters illustrates the type of corrosion experienced and the comparative rates of corrosion of the engineering metals and alloys. Cast iron, alloy and plain steels, copper, aluminum, bronze, and monel metal were attacked. It was concluded that all metals that form sulphates cannot be used for air-preheater plates without corrosion and eventual plate failure.

Similar results have been found with tubular recuperative air heaters.[35] Berk[36] has reviewed British experience, and Henning and Rögener have reported German difficulties with economizers.[37]

Several steps have been taken toward practical solutions of the problem. In economizers, the corrosion is prevented by elevating the feedwater temperature. In air preheaters, the inlet air is first warmed by recirculating with it part of the preheated air. The air warming might also be done with indirect heat from exhaust steam or in a cheaper expendable air heater.

An automatic means for regulating the air-inlet temperatures to preheaters so as to prevent dew point conditions would be highly desirable. A practical and reliable dew point indicator for dusty flue gas would be the first step toward this objective.

Addition of dolomite ($CaCO_3$-$MgCO_3$) to a 2.75-pct-sulphur fuel oil by the Florida Power Corp. is reported to have lowered the dew point from 250° to 275°F down to 150°F.[38] Loose calcium sulphate ($CaSO_4$) was formed. The effect of dolomite on air-heater corrosion has not yet been determined, either for oil or coal. The effect on SO_3 in flue gas of

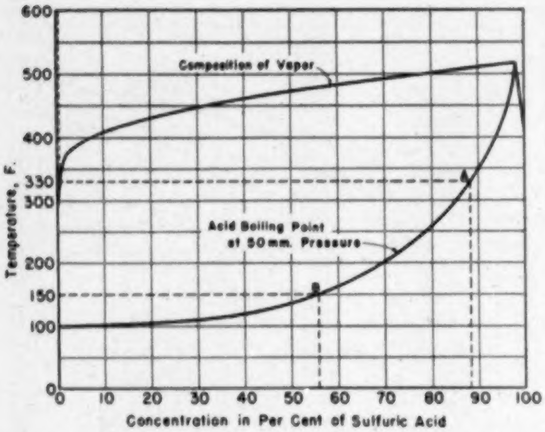

Fig. 6—Boiling point-vapor composition curve for sulphuric acid solutions at 50 mm Hg pressure. (Courtesy H. F. Johnstone).

the natural lime in coal ash or limestone additive to coal is also apparently unknown.

Substitution of special glass tubing for metal in the condensation region of air heaters is being tried, as is a method for periodic replacement of short lengths of corrodible tubing.[38] However, the clogging of the tubing by acid-trapped dust would likely still be a problem. This leads to the suggestion that high-efficiency dust collectors be installed ahead of the air heaters.

Where spreader stokers are used, the cinders carried through the air heater would help to scour off deposits. Where that is the case, dust collectors may well be located after the air heaters.

Cooling Flue Gas Below Dew Point: Future opportunities and possible developments are not being overlooked. An interesting and possibly promising line of attack would be deliberately to produce a maximum of SO_3 with a view toward condensing it out with the ample quantities of water vapor already present in flue gas. The air preheater to condense the acid would have to be of acid-proof construction. Instead of discharging the flue gas at 330°F as in the example,[4] heat might be extracted down to an arbitrary temperature of 150°F. Table V lists the sources and amounts of recoverable heat for this installation. Only a small fraction of the water vapor in the flue gas would be condensed.

Table V. Recoverable Heat in Flue Gases Between 330°F and 150°F

Source of Heat	Lb Per Lb Coal	Btu Released Per Lb Coal	Btu in Coal, Pct
SO_2 to SO_3 reaction	0.082	42	0.34
H_2O sensible heat	0.494	43	0.35
H_2O latent heat of condensation	0.044	44	0.36
$H_2O + SO_3 = H_2SO_4$ (80 pct strength, approx.)	0.127	31	0.25
Dry gases	11.806	496	4.00
Total		656	5.30

The hypothetical change in the heat balance for the boiler is given in Table VI.

Even if only half the sulphur in the coal were converted to SO_3 and the gases were cooled to 150°F, additional heat recovery would be 4.83 pct.

The weight of raw acid may be sufficient to wash away the dust trapped, especially if the acid were recirculated for that purpose.[39]

If the SO_3 in normal flue-gas concentration were removed from the flue gas by a suitable dry reagent and the gases were then cooled to 150°F, no condensation of moisture would occur and no acid would be formed. The added heat recovery would be 4.35 pct. Where flue gas is presently cooled to 250° to 275°F, the opportunity for gain would be proportionally less. Some large boilers now have average operating efficiencies of 89 to 90 pct.

These and other possibilities are under study to increase heat recovery. The increased heat transfer surface for air preheating will add to the investment of the boiler, but the heat recovery should be sufficient to establish the economic balance at a lower flue-gas temperature than is now practical.

Air Pollution Control: No study of this kind would today be complete without considering the effect of combustion methods and heat-transfer equipment on air pollution. Smoke and soot have fortunately been reduced to low levels by better combustion. Fly-ash emission is being brought under control by dust collectors and precipitators, but more needs to be done. Economical removal of sulphur oxides from flue gas has been investigated for years by Johnstone in the United States and by others abroad.[39] Flue-gas washing has been practiced in a few plants,[40] but the disposal of the effluent water created serious problems.

By further cooling of the flue gases to recover heat, it may be possible to combine the removal of at least some of the sulphur oxides and more of the flue dust.

Table VI. Heat Balances in Pct of Btu in Coal

Stack Gas Temperature	330°F		150°F
Heat absorbed by steam-generating unit	85.69		90.99
Heat loss in dry chimney gases	5.47		1.47
due to moisture in air	0.14		
due to moisture in coal	0.43	4.64	3.93
due to combustion of hydrogen	4.07		
due to unconsumed carbon	2.00		2.00
due to radiation and unaccounted	2.20		1.61
	100.00		100.00

An interesting sidelight is the beneficial effect of SO_3 on the collection efficiency of electrical precipitators[41] through lowered electrical resistivity of the fly ash in the presence of low percentages of SO_3.

Experimentation with new principles and chemical processes which combine recovery of heat and air-pollutants may lead to an economical answer.

Limitations of Coal Cleaning: The misbehavior of ash and sulphur in boilers naturally suggests consideration of the advantages and economics of coal cleaning. As is well known, the sulphur and ash contents of coals can be reduced by density separation of particles of higher and lower ash contents near a selected specific gravity. Modern coal preparation plants improve coal quality by reducing the ash and sulphur percentages, and by producing a more uniform product. The effect on the ash-softening temperature varies. Depending on the coal, cleaning may raise or lower the ash-softening temperature.

Better coal preparation has been the trend and many users can attest to the benefits. However, there are definite physical and economic limitations to coal cleaning. The present cost of coal preparation for steam generation, which generally ranges between 35¢ and 75¢ a ton of product coal, permits

removal of only the free impurities, plus possibly heat drying the finer sizes of washed coal.

A number of coal producers have investigated the cost of further reducing ash contents beyond present practical limits. The process involves progressively crushing the coal to release particles high in ash content and cleaning at lower specific gravities. The yield of final product coal is reduced by the more intensive cleaning (separation), as more tonnage and Btu's are rejected to gob. Two examples of confidential origin may be cited in Table VII.

Table VII. Cost Studies of Raw and Washed Coals at Two Mines

Eastern Bituminous Coal		Midwestern Bituminous Coal	
Ash Content, Dry Basis, Pct	Cost* Per Ton of Product	Ash Content, Dry Basis, Pct	Cost* Per Ton of Product
15.5 raw	\$3.50**	16.5 raw	\$3.50**
8.5	4.40	6.5	4.75
7.5	4.75	6.0	4.80
6.5	5.25	5.5	4.90
5.5	6.05	5.0	5.20
		4.5	5.70
		4.0	6.90
		3.5	8.50

* Includes washing and loss of coal.
** Arbitrary.

The rapidly rising cost per ton of cleaned coal is evident for both coals. The difference in cost between the two coals reflects the greater difficulty of cleaning this particular eastern coal as compared with the midwestern coal.

The reduction of sulphur, iron, and alkali contents of the coal is even more difficult and more impractical by methods known today than is the reduction of ash. Free marcasite and pyrite (FeS_2) particles of macroscopic size (density = 5) are readily removed in washing because of their high density as compared with coal (1.3 to 1.5). However, part of the sulphur and iron is thoroughly distributed in coal as sulphate and as finely disseminated pyrite particles of microscopic size. Sodium and potassium are present as chlorides. They are distributed throughout the coal and the shaly matter.

Because of the technical and economic limitations on coal cleaning by known processes, boiler and combustion-equipment designs have been developed to cope with the ash and sulphur problems. In this way the most favorable overall cost balance can be struck on the basis of cost of steam.

Beyond the cost of Btu's delivered, the economic value of well-prepared industrial steam coal for large modern steam generators is a subject for evaluation by each coal consumer and coal source. Ash increases the coal tonnage to be handled and pulverized. Boiler operating labor and maintenance increase with ash content of the coal. When ash deposits and slagging force a reduction in boiler load or shutdown, the loss could have paid for much coal preparation, particularly if the boiler load must be shifted to older and less efficient peak-load equipment. Complete information is lacking, for even one boiler plant, to make possible a comparison of cost of operation and maintenance for two coals of different ash contents and different characteristics.

Boiler Size and Cost as Related to Fuel Characteristics: Boilers are designed to fit a fuel specification. Boilers designed for coals of high ash-softening temperature can be more compact than those designed for coals of low ash-softening temperature. Larger furnaces are required for the latter coals to reduce the temperature of the gases entering the first convection bank 150°F below the ash-softening temperature. The tube spacing must also be larger in the first convection zone to prevent bridging of ash deposits.

The design and selection of boilers is a teamwork job between boiler manufacturer and utility purchaser. Some select more conservatively than others. After the boiler is installed, the operating crew must learn the characteristics of the boiler and the control of combustion and ash.

The magnitude of the difference in cost between boilers for different fuels was investigated with the boiler manufacturers. The purpose of the inquiries was to determine the cost of liberal boiler sizing as one way of reducing the tight ash deposits on boiler surfaces.

During an interview, one manufacturer prepared estimates for a 900,000-lb-an-hr steam generator, 1850 psi, 1000°F. Included were an outdoor-type boiler, superheater, economizer, air heater, soot blowers, pulverizer, coal feeders, dust collector, but not feedwater pumps, fans, or walkways. The figures are summarized in Table VIII.

The increased price for a boiler to consume coals of low ash-softening temperature was 7 pct. The increase in cost was primarily due to the increased size of furnace and convection passes and extra heat-exchange surfaces required. The estimate for an "ashless" coal-fired boiler is based on experience with natural gas, except that the cost of coal feeders and pulverizer was included, but not dust collector or soot blowers.

Table VIII. Relative Prices of Boilers and Auxiliaries

Type of Bituminous Coal	Price of Unit, Pct	Relative Price, Pct
"Ashless"	100	
Ash of AST 2750°F	131	100
Ash of AST 2100°F	140	107

In 1944 the Fairmont Coal Bureau[60] published a similar analysis made by another boiler manufacturer, see Table IX.

Table IX. Relative Prices of Boilers and Auxiliaries, Fairmont Coal Bureau

400,000 lb steam per hr — 950 psi, 850°F
Efficiency: 87 pct — Capacity factor: 85 pct
Pulverizer firing, bituminous coal

	For Coal of AST 2550°F	For Coal of AST 2070°F
Furnace:		
Width	21 ft 10 in.	21 ft 10 in
Height	34 ft 6 in.	42 ft 6 in.
Depth	20 ft 1½ in.	24 ft 0 in.
Volume, cu ft	19,900	28,000
Additional furnace volume cost		\$22,000
Additional building volume		30,000 cu ft
Additional building cost		\$15,000
Total additional cost		\$37,000
Total cost of the boiler and auxiliaries	\$368,000	\$390,000*
Pct increase in cost of boiler only		6

* 60 pct of 1950 price.

The increased price of 6 pct for a boiler to burn low-AST coal compares closely with that of 7 pct obtained independently in the first example.

The additional investment in the larger boiler must be amortized by savings in cost of Btu's in the lower-fusion coal. The effect on the price of coal can be illustrated with conservative example:

Assumptions:
- Boiler of 425,000 lb per hr steaming capacity.
- Cost installed: $1,000,000 for high-AST coal.
- Additional cost of $100,000 for low-AST coal.
- Coal: 11,500 Btu per lb.
- Evaporation: 9.42 lb water per lb coal.
- Load factor of 80 pct, availability 95 pct.
- Avg load = 425,000 x 0.80 = 340,000 lb an hr.

Annual coal consumption = 340,000 x 0.95 x 8760/(9.42 x 2000) = 150,000 tons.

Amortization of increment investment over 6 yr: $100,000/6 = $16,700 a year.

Amortization rate = $16,700/150,000 = $0.11 per ton of coal.

The additional cost per ton of coal is so low that it is now common practice for utilities to increase their fuel flexibility by installing boilers designed for a wide range of coals. The increment cost in a larger boiler is less than any known preparation or treatment of a low-AST coal to render it suitable for the smaller boiler. A boiler manufacturer advised that the boiler should be made to suit the worst conditions, that the furnace should be built large enough, and that gas temperatures are lower with a clean furnace.

As indicated earlier, other ash and sulphur factors must also be considered. Operating labor increases with the quantity of ash handled. A larger tonnage of high ash coal must be unloaded, stored, and conveyed for equal steam output. Increased maintenance on coal and ash conveyors, higher cost of ash removal, more frequent soot blowing, and higher pulverizer wear are among the costs that must be considered.

Future Possibilities and Conclusions

1—The designs of modern large power boilers and combustion devices are highly advanced for thermal efficiency and for capacity to cope with ash in coal.

2—Further research on ash behavior and slag prevention would be desirable. Improved soot blowers, pellet guns, tube vibrators, and operating techniques now under development will also assist in keeping boiler tubes clean. The manifold advantage of tube curtains in their various forms will help reduce furnace temperatures and adherent deposits.

3—The role of non-sulphate ash dust in condensing the alkali sulphates and rendering them nonadherent to tubes should be investigated with a view to wider practical application.

4—Cooling of flue gases to 150°F is a worthy objective provided that corrosion and deposits in air preheaters can be prevented. Each 35° to 45°F reduction in flue-gas temperature increases boiler efficiency by 1 pct.

5—Several hypothetical possibilities exist for the control of SO_3 in flue gas: SO_3 formation might be prevented by techniques to be developed, but the likelihood is slight. Normal amounts of SO_3 in flue gas might be absorbed on basic dusts ahead of or in the air preheater. SO_3 could be condensed out with water vapor by cooling of gases in acid-proof preheaters.

6—SO_3 now assists in reducing air pollution from power plants by reducing electrical resistivity of fly ash in electrical precipitators. In future air preheaters of acid-proof construction, the sulphuric acid produced by SO_3 and water vapor may be useful in trapping flue dust. The acid could be recirculated if necessary to wash the dust off the surfaces. It may become advantageous to convert as much as possible of the sulphur to SO_3.

7—Economic and technical opportunities for solving the ash and sulphur problems with coal appear to be more promising at present through combustion and utilization in power boilers than through supercleaning of coals beyond the present economic limits.

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A Physical Explanation of the Empirical Laws of Comminution

by D. R. Walker and M. C. Shaw

The laws of Kick and Rittinger are explained as functions of particle size with metal cutting theory. Comminution is shown to be basically the same process as metal grinding. The machine shop type of grinding operation is used to study mineral crushing. Evidence indicates that plastic flow occurs in comminution.

THE laws of comminution of Kick and Rittinger have been debated for many years. Certain data obtained from ball mill and drop tests are found to be in approximate agreement with Rittinger's law while other data would seem to support Kick's law. In all these tests the energy required to produce a wide variety of particle sizes has been studied. Surface area has also been measured with considerable difficulty, at first because it was thought that the energy associated with crushing was that required to create new surface, later because of precedent and because a more fundamental variable was unknown.

Tests made with the machine shop type of grinding process show that normally brittle materials that are usually crushed can behave like the more ductile metals. Brittleness is then a relative property largely dependent on specimen size and stress state. The machine grinding operation offers a convenient means for studying the comminution of materials in which the important size variable retains its identity. The energy required to grind a certain volume of material is shown to be strongly dependent on size. Kick's law is found to hold very well for particles below 1 micron in size, while Rittinger's law holds approximately in the comminution of larger particles. While a physical origin of Kick's law is established in terms of the theoretical strength of materials having a perfect lattice structure, there is found to be no physical basis for Rittinger's law.

Because of its industrial importance, comminution has commanded considerable technical attention and has been the subject of continuous study for nearly a century. A number of empirical rules have been proposed to make it possible for grinding energies to be predicted. Of these so-called laws of comminution those bearing the names of Kick and Rittinger have been most widely applied and debated.

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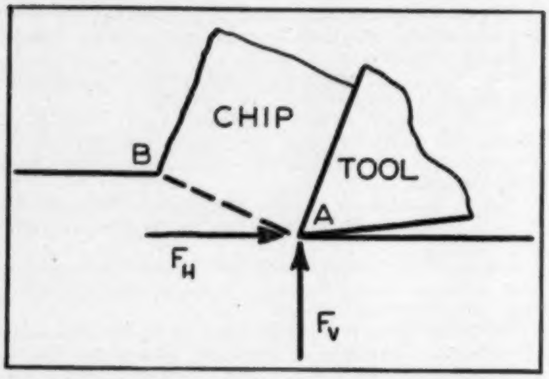

Fig. 1—Schematic representation of metal cutting operation.

Kick's law is based upon a critical strain energy concept. It states that the energy required to fracture a specimen is directly proportional to its volume and independent of the number or size of particles into which it is reduced. Stated algebraically this is

$$U = K_1 V \qquad [1]$$

where U is the energy to which the volume V must be subjected to cause fracture and K_1 is a proportionality constant.

The law of Rittinger is based upon the prediction that the energy required to crush a substance is that needed to produce the new surface generated. Or,

$$U = K_2 (A_2 - A_1) \qquad [2]$$

where U is the energy required to crush a material when the change in area involved is $(A_2 - A_1)$, and K_2 is a proportionality constant.

Investigators using Rittinger's law have found that crushing efficiencies are generally extremely low. Some data of Martin[1] who crushed quartz in an 18 x 18-in. tube ball mill using 1-in. diam balls are shown in Table I.

Table I. Representative Ball Mill Data

Energy Consumed, U Ft-Lb	New Area Produced (A_2-A_1) Ft²	$U/A_2-A_1 = K_2$ Ft-Lb Per Sq Ft
243,375	3,971	61.3
470,250	7,852	59.9
699,190	11,170	62.6
892,346	14,941	59.7
1,097,300	17,899	61 3
		Average 60.9 ft-lb per sq ft = 5.07 in-lb per sq in.

The data of Table I are seen to be in good qualitative agreement with Rittinger's law, but in poor agreement with Kick's law. However, the surface energy of quartz is known to be about 1000 ergs per sq cm, which is 0.0057 in-lb per sq in. in English units. When this value is compared with the mean value of K_2 in Table I, it is found that only 0.112 pct of the energy associated with the process is being used to supply the necessary surface energy. Thus from Rittinger's point of view the efficiency of Martin's ball mill is but 0.112 pct, which is remarkably low for any engineering process.

Much of the literature in the field of comminution is found to consist of arguments supporting or rejecting one or the other of the two laws which have been mentioned here. Some investigators have found that their data tends to obey Kick's law, while others have gathered evidence which they use emphatically to defend Rittinger's law. Haultain[2] in 1923 and more recently Bond[3] have suggested that the actual law lies half way between those of Kick and Rittinger.

Experimental work in the field of comminution has been subject to several difficulties. In ball-milling experiments the total energy consumed in crushing a charge from one mean particle size to another has been measured. This energy is the energy required to crush particles of a wide variety of sizes. From such data it is not possible to determine precisely the energy required to crush particles of any one size. The chief variable studied has been the surface area, determined in the past by elaborate screening analyses and more precisely in recent years by means of gas adsorption and permeability techniques.[4] In view of the very small percentage of the energy of comminution associated with the development of new surface, it would appear that entirely too much emphasis has been placed on surface area in previous investigations.

Drop tests have been adopted in some studies to improve the experimental precision. However, interpretation of the data is still made difficult by the variety of particle sizes that result in such tests. It would appear that a substantially different experimental approach is needed to develop further the concepts of comminution.

In addition to comminution there is a second grinding process practiced on a wide industrial scale. This is the type of grinding done in the machine shop, where the objective is to produce a smooth, dimensionally accurate surface. Unlike the powders produced in comminution, the small chips formed in machine grinding are not of value and hence are usually discarded. Even though the objectives in comminution and machine grinding are diametrically opposed, there is considerable similarity between the two processes.

While it might be argued that comminution is concerned with materials normally considered brittle, whereas machine grinding involves metals having appreciably greater ductility, it will be shown later in this paper that this difference is actually more apparent than real. Since the intention is to apply to the comminution process techniques developed in the study of metal cutting, it may be well to review briefly the manner in which a metal behaves

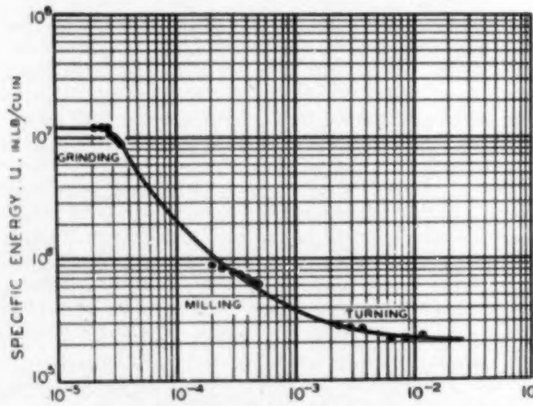

Fig. 2—Variation of specific energy with particle size for SAE 1112 steel.

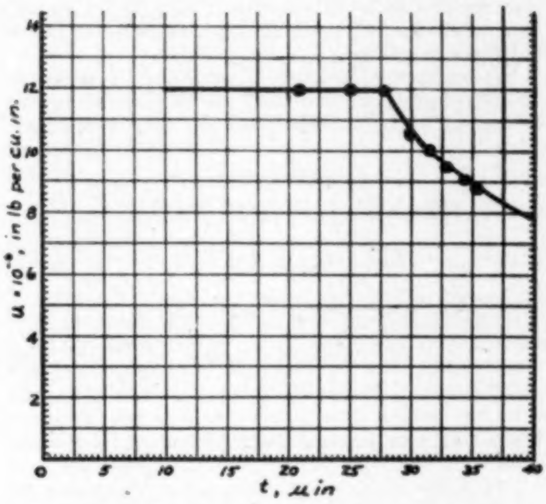

Fig. 3—Variation of specific energy with particle size in grinding for SAE 1112 steel.

when it is cut, before continuing the discussion of comminution.

Metal cutting is a shear process in which a chip is caused to flow plastically when a tool is advanced across the workpiece. The action is as shown in Fig. 1. The material below line *AB* (the shear plane) is plastically undeformed, while the material in the chip above *AB* is deformed uniformly. All the shear strain is confined to a very narrow zone extending along *AB*. As the metal crosses line *AB* it is subjected to a relatively large and rapidly applied shearing strain.

The calculation of the energy consumed per unit volume of metal cut or specific energy is one of the parameters which has proved to be very important in metal cutting studies.[5] When the depth of cut, t, taken by a tool, is decreased, the specific energy, u, is found to increase. The curve shown in Fig. 2 which covers an extremely large range of size was recently obtained[6] by use of a tensile test and three different cutting operations: turning, very high speed milling, and grinding. The material in all these tests was SAE 1112 steel.

The grinding end of Fig. 2 is shown to a larger scale in Fig. 3. Here it is evident that the energy per unit volume remains constant as the depth of cut is increased to a point, but then proceeds to decrease with further increase in depth of cut. The depth of cut or layer thickness corresponding to the break in the curve is about 27 micro in. or ¾ of a micron. It is of interest to this discussion to consider the cause for the sudden break in the energy curve of Fig. 3.

When an energy balance is made for a cutting operation it is found that practically all the energy is associated with the shear and friction processes. As in comminution, the surface energy involved in the generation of new surface is negligible, and energy associated with change in velocity of the metal as it crosses the shear plane also can be neglected.

$$u = u_s + u_f \qquad [3]$$

where u is the total energy per unit volume of metal cut, u_s is the shear energy per unit volume, and u_f the energy per unit volume required to overcome friction on the tool face. It has been shown in a recent paper[6] that in fine grinding operations u_s and u_f are about equal, while the shear strain in the chip is about 3. Since in plastic flow the shear energy per unit volume (u_s) is equal to the product of the average shear stress and the shear strain, then

$$u \cong 2\,u_s \cong 2\tau\gamma \qquad [4]$$

Since γ is approximately 3

$$\tau \cong \frac{u}{6} \qquad [5]$$

The value of u for the flat portion of Fig. 3 is seen to be about 12×10^6 in.-lb per cu in. and the corresponding value of shear stress is thus found to be about 2×10^6 psi from Eq. 5.

In Fig. 4 a true stress-strain tensile curve is shown for the material of Figs. 2 and 3, together with the stress-strain points for the cutting tests. In plotting the cutting points shear stress and shear strain were converted to normal stress and strain coordinates by use of the maximum shear theory. From Fig. 4 it is evident that the stresses required for cutting increase significantly as the depth of cut is decreased. The large variation in specific energy that was observed to occur with specimen size, Fig. 2, is thus seen to be due to a corresponding change in the flow stress of the material with specimen size. While the flow stress in shear is but about 50,000 psi for the tensile test specimen of 0.505 in. diam, the shear flow stress in grinding is found to be about 2×10^6 psi when the particle size is less than 27 micro in.

For a long time physicists have held that metals should exhibit much higher values of flow stress and strength than they actually do. The strength of a metal specimen is found to be $G/2\pi$, where G is the elastic shear modulus of the metal, in one approximate calculation[6] based upon the forces required to slide one layer of atoms over another in a material having a perfect lattice structure. For steel, G is about 11.5×10^6 psi, and hence the theoretical shear strength of a steel specimen with no lattice imperfections should be about 1.8×10^6 psi, i.e., $\frac{11.5\times10^6}{2\pi}$.

This value is seen to be in good agreement with the 2×10^6 psi observed for the maximum shear stress in fine grinding. It would thus appear that the specific energy (and hence the shear stress) remains constant as long as the depth of the layer removed is less than that corresponding to the spacing of the inhomogeneities that are found in all metals. At a depth of cut of about ¾ microns the probability of encountering an inhomogeneity becomes noticeable and from this

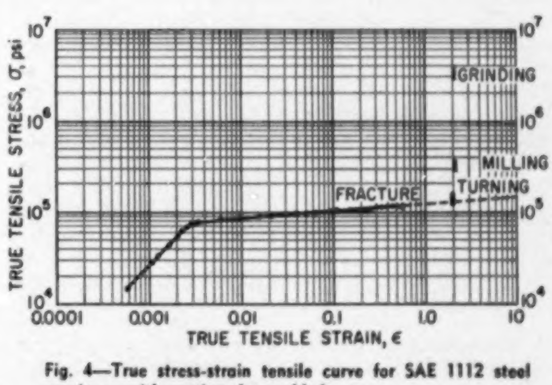

Fig. 4—True stress-strain tensile curve for SAE 1112 steel specimen, with cutting data added.

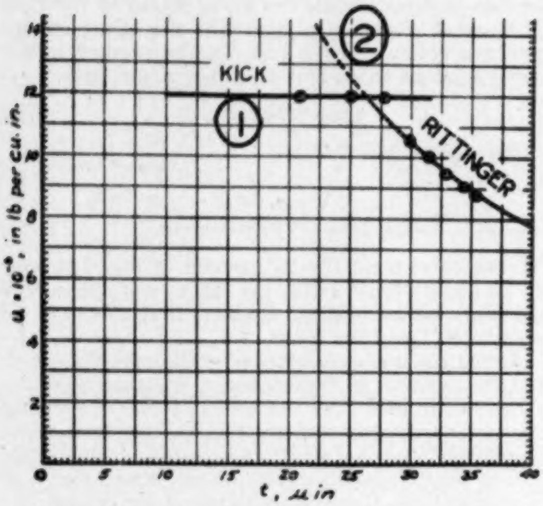

Fig. 5—Relation between grinding data of Fig. 3 and equations of Kick and Rittinger.

point on the flow stress of the material decreases markedly with an increase of the depth of the layer removed and hence the probability of finding an inhomogeneity.

From x-ray diffraction studies on metals, the size of the crystallites within which no lattice imperfections are found has been estimated to be about 10^{-4} cm (1 micron). This value is seen to be in good agreement with the thickness of the layer at which the effect of inhomogeneities become noticeable in metal cutting. Furthermore, the energy per unit volume is found to be independent of the hardness of the metal cut when the depth of cut is below the critical value of about ¾ micron, but it is found to be very important for larger cuts. This observation is in agreement with the foregoing picture. If the theoretical strength of the metal is $G/2\pi$, it should be unchanged by hardness, since G is known to be independent of hardness. Thus the shear stress and energy per unit volume in a cutting operation with a chip size less than about 27 micro in. should be independent of hardness.

From the brief review of recent metal cutting research that has been presented here it is evident that the size of the particle removed is an extremely important variable. The shear stress associated with the removal of a particle is found to increase significantly with decreased particle size, owing to the decreased probability of finding weakening inhomogeneities with decreased size. When a point is reached, in the reduction of size, where essentially no inhomogeneities are encountered, the strength of the metal remains constant with further reduction in size. The shear energy per unit volume is found to reflect directly changes occurring in shear strength.

The laws of Kick and Rittinger will now be interpreted in terms of the metal-cutting data that have been presented. If both sides of Eq. 1 are divided by the volume of the particle, then

$$u = \frac{U}{V} = K_1 \qquad [6]$$

which states that the energy per unit volume is a constant. This is seen to be in agreement with the horizontal portion of Fig. 2, and hence Kick's law may be said to be in agreement with metal cutting data when the depth of the layer removed is below the critical value.

In expressing Rittinger's equation in terms of specific energy it will first be observed that no change in volume accompanies any plastic deformation. Hence,

$$V_1 = V_2 = V \qquad [7]$$

If both sides of Eq. 2 are then divided by V,

$$u = \frac{U}{V} = K_2\left(\frac{A_2 - A_1}{V}\right) = K_2\left(\frac{A_2 - A_1}{V_2 - V_1}\right) \qquad [8]$$

This is seen to be the definite integral of

$$\frac{du}{d\left(\frac{A}{V}\right)} = K_2 \qquad [9]$$

When this equation is integrated

$$u = K_2\,\frac{A}{V} + u_o \qquad [10]$$

is obtained where u_o is the integration constant. However, as indicated in Eq. 11,

$$\frac{A}{V} = \frac{1}{t} \qquad [11]$$

V varies directly with the product of A and t where t is a mean thickness of the layer removed. Hence, from Eq. 10 and 11

$$u - u_o = \frac{K_2}{t} \qquad [12]$$

In Fig. 5 the data of Fig. 3 are shown replotted, and curve *2* is drawn in accordance with Eq. 11, the constants being:

$u_o = 200{,}000$ in.-lb per cu in.
$K_2 = 300$ in.-lb per sq in.

This curve is seen to be in good agreement with the grinding data. However, when this same curve, *2*, is plotted with all the cutting data, see Fig. 6, the agreement is not quite as good. The milling data would appear to be in better agreement with curve *3* having constants

$u_o = 416{,}000$ in.-lb per cu in.
$K_2 = 103$ in.-lb per sq in.

From this comparison of the laws of Kick and

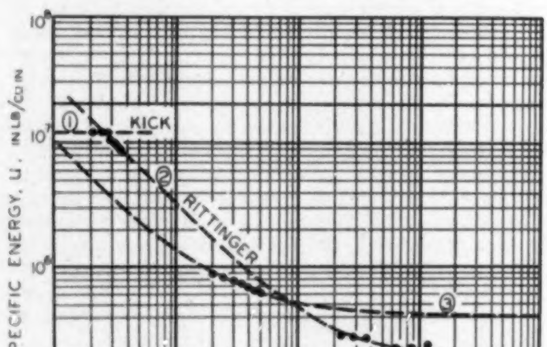

Fig. 6—Relation between cutting data of Fig. 2 and equations of Kick and Rittinger.

Rittinger with metal cutting data it is evident that Kick's expression

$$U = 11.95\ V, \text{ in.-lb} \qquad [13]$$

is in good agreement with cutting data when the depth of cut is less than about ¾ micron, while Rittinger's expression

$$U = 300\ \Delta A, \text{ in.-lb} \qquad [14]$$

is in good agreement with the grinding data, while the expression

$$U = 103\ \Delta A \qquad [15]$$

is in better agreement with the milling data than Eq. 14. However, these expressions are not in such good agreement with the experimental data for the reasons originally given by their authors. Kick's law applies because the material exhibits full theoretical strength and not because of any correctness of a strain energy concept. Rittinger's law holds despite the fact that the surface energy on which it is based is completely negligible. It is only because of the fact that a simple functional relationship exists between the size of a particle and its surface area that the Rittinger expression can be written in terms of area. The particle size is obviously the important variable, and Rittinger's expression should preferably be written

$$u - 200{,}000 = \frac{300}{t} \text{ in.-lb. per cu in.} \qquad [16]$$

in as much as the emphasis is then upon the physically important variable, t, rather than the physically unimportant variable, ΔA, as in Eq. 13.

Mineral Grinding

To investigate any differences that might be observed between the machine grinding of ductile metals and materials normally considered brittle, tests similar to those already described for metals were performed on minerals. The detailed test procedure is to be found in reference.[4] Some grinding tests were run using a 46-grit 8-structure aluminum oxide wheel, while others were made using a 24-grit aluminum oxide abrasive belt. The belt made it possible to remove chips of greater size than could be handled by the wheel.

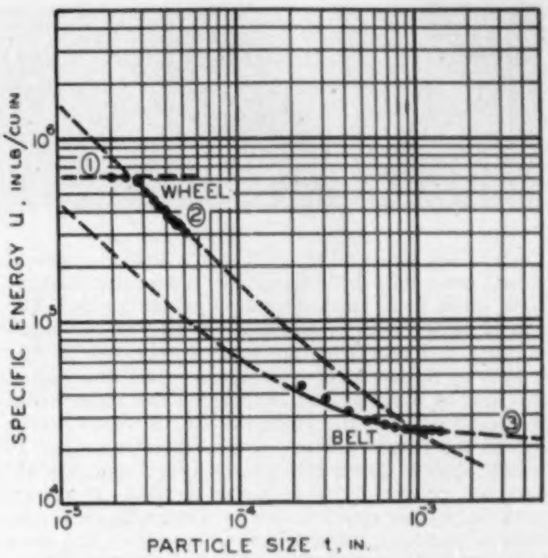

Fig. 8—Variation of specific energy with the particle size for gypsum.

Energy curves for marble, gypsum, and talc are given in Figs. 7 to 9. There is a striking resemblance between these data and the data for SAE 1112 steel given in Fig. 6. There is a region, *1*, for which the energy per unit volume is constant, followed by a region where the energy per unit volume decreases with depth of cut. It was found in each case that a curve in agreement with Eq. 12 (Rittinger's law) could be passed through the grinding-wheel data or the belt-grinding data, but that the same curve did not hold for both of these regions. This was also found to be the case with the metal, as may be seen in Fig. 6. This result simply means that while Rit-

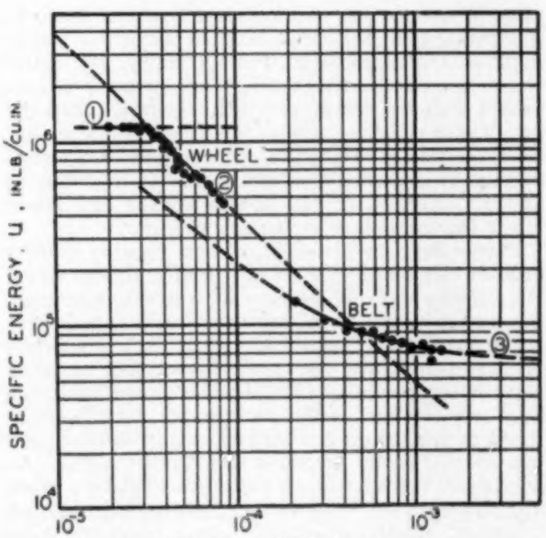

Fig. 7—Variation of specific energy with the particle size for marble.

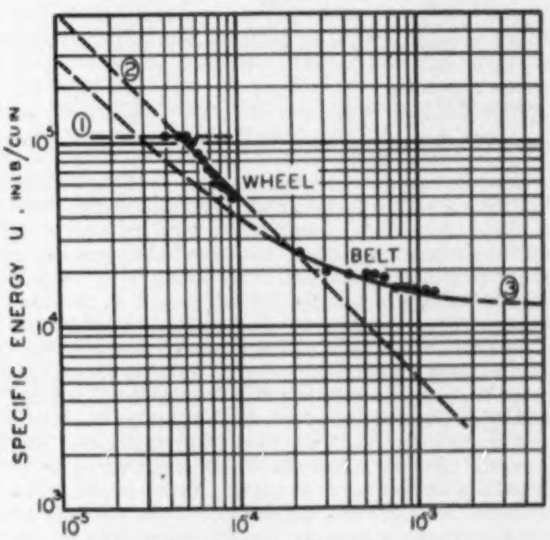

Fig. 9—Variation of specific energy with the particle size for talc.

Fig. 10—Impressions made in marble with a diamond pyramidal indentor under loads of (a) 2.15 g, (b) 10 g, and (c) 5 g X 1200.

tinger's law holds quite well over a limited range of size it must only be regarded as a first approximation when it is applied over a wider range. The constants for energy equations 6 and 12 are summarized in Table II for the tests of Figs. 7 to 9. The values of K_s are seen to decrease while those of u_s increase in a systematic way along the range from the small particle size characteristic of wheel grinding to the larger particle size of belt grinding. It would appear that the Rittinger expression, Eq. 12, does not predict quite as much increase in energy with decreasing particle size as is actually observed. This is evident in Figs. 7 to 9 where curve *3* is seen to be below curve *2* for small depths but above curve *2* for large depths.

The constant K_s may be interpreted as the energy required to generate a unit quantity of new surface. This quantity is seen to vary not only with the material ground but also with the size of the particles removed. For example, the value of K_s for marble is approximately 39.3 in.-lb per sq in. for the size range extending from 30 to 90 μ in., while K_s equals about 16.2 in.-lb. per sq in. for the size range from 200 to 2000 μ in. Since the surface energy for marble is about 0.006 in.-lb. per sq in., it is again evident that the surface energy represents a very small percentage of the energy required to generate a unit of new surface.

Table II. Energy Constants for Mineral Grinding

			Region 2		Region 3	
Material	K_1 In.-Lb Per Cu In.	Critical Size μ In.	u_s In.-Lb Per Cu In.	K_s In.-Lb Per Sq In.	u_s In.-Lb Per Cu In.	K_s In.-Lb Per Sq In.
Marble	1.20 x 10^6	33	8500	39.3	59000	16.2
Gypsum	0.62 x 10^6	25	9100	15.3	20600	4.3
Talc	0.11 x 10^6	50	0	5.0	12200	2.76

Some investigators have reported that Kick's law holds best when particles of large size result from comminution. Figs. 7 to 9 show that the curves of specific energy vs particle size tend to become flat in the region of large particle size. Rittinger's law and Kick's law will thus give similar results in this size region. The tendency toward constant specific energy for comminution of large particles is to be expected since the larger the particle size becomes, the more the particle behaves like the material in bulk, and the specific energy of fracture thus tends towards that for fracturing the bulk material. This loss of size effect when the specimen becomes quite large is also observed in metals. Tensile tests made with metals show no dependence upon the size of the specimen as long as the specimen diameter is larger than about 0.1 in.

Such materials as marble are normally considered to fracture without undergoing previous plastic strain. Materials that behave in this way are said to be brittle. If marble is ground in the region in which Kick's law holds (u = constant), the shear stress developed at rupture would be the theoretical value, $G/2\pi$, where G is the shear modulus. If the material were perfectly brittle it would behave elastically to the point of rupture and from Hooke's law ($\tau = G\gamma$) the shear energy per unit volume involved would be

$$u_s = \frac{1}{2}\tau\gamma = \frac{1}{2}\frac{\tau^2}{G} \qquad [16]$$

Substituting $G/2\pi$ for τ,

$$u_s = \frac{G}{8\pi^2} \qquad [17]$$

Now, the shear modulus for marble is about 3.15 x 10^6 psi and hence the theoretical strength and elastic shear energy per unit volume would be as follows:

$$\tau = \frac{G}{2\pi} = 0.5 \times 10^6 \text{ psi}$$

$$u_s = \frac{G}{8\pi^2} = 0.04 \times 10^6 \text{ psi}$$

When this specific elastic shear energy is compared with the total energy per unit volume observed in the Kick's law region for marble (1.2x10^6), it would appear that

$$\frac{1.2 - 0.04}{1.2} \times 100 = 97 \text{ pct}$$

of total energy was consumed in friction. Since in the grinding of steel only about 50 pct of the energy input is associated with friction, the 97 pct figure would seem to be far too high, and hence it is suspected that marble is not really brittle under the grinding conditions investigated here, but rather flows plastically to a considerable extent before rupturing. This plastic flow would tend to increase the amount of shear energy in the process and hence lower the percentage of friction energy.

Plastic flow in grinding or any type of cutting process can be carried to far greater strains without rupture than in the case of a tensile specimen. For example, the strain to rupture in a true-stress-strain tensile test is about 0.5, while the corresponding strain in cutting the same material will be 2.0 or more without rupture, Fig. 4. The reason for these large strains lies in the fact that a large normal stress is present on the shear plane in cutting, and this normal stress increases the rupture stress. As previously mentioned, all materials contain a great many minute cracks and other inhomogeneities, and a normal stress on a shear plane prevents these cracks from growing to cause rupture. A material that contains large flaws will rupture at a lower stress level than one that contains smaller flaws.

Therefore, the strength of a small specimen will exceed that for a large specimen, since the probability of finding a flaw of a given size will be less in the small specimen. There are thus two important reasons why materials in grinding should be capable of withstanding much higher stresses without fracture than in bulk: 1—because of their small size and 2—because of the presence of a large normal stress on the shear plane.

When these two elements are present, materials that normally behave in a brittle manner in bulk may exhibit considerable ductility. For example, marble is usually considered to be a brittle substance, since it exhibits low tensile strength in bulk and essentially no plastic deformation before rupture. However, it is possible to cause extensive plastic strain in marble if the specimen is small and the pressure high. In Fig. 10 three impressions are shown that were made in a polished marble surface by a diamond pyramidal indentor. A different load was used in each case. In the first and second impressions a good plastic impression of the diamond is seen but there is some evidence of cracks extending from the corners of the impression. In the case of the third impression plastic flow has occurred, but no cracks are resolved at a magnification of 1200 x. This example illustrates that brittleness is a quantity that has meaning only when the size of the specimen is specified. Furthermore, it might be said that a material will behave in a brittle manner when its size is such that it contains cracks of sufficient size to preclude any measurable plastic deformation before fracture. Glasses, whether in the annealed or strained state, contain a large number of imperfections which vary widely in size; hence glasses are brittle over a wide range of size.

The Vickers hardness values (load in kilograms divided by projected area of impression in square millimeters) corresponding to the impressions of Fig. 10 are as follows:

Load, Gm	Vicker's Hardness, Kg/mm²
15	400
10	417
5	475

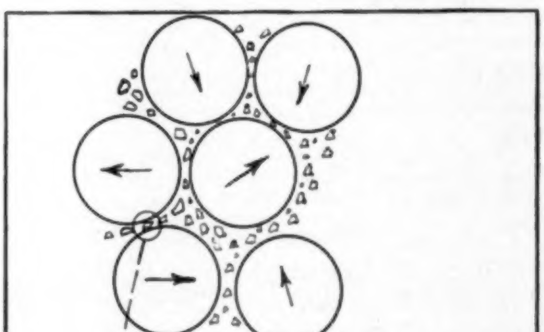

Fig. 11—Schematic representation of a particle being crushed in a ball milling operation.

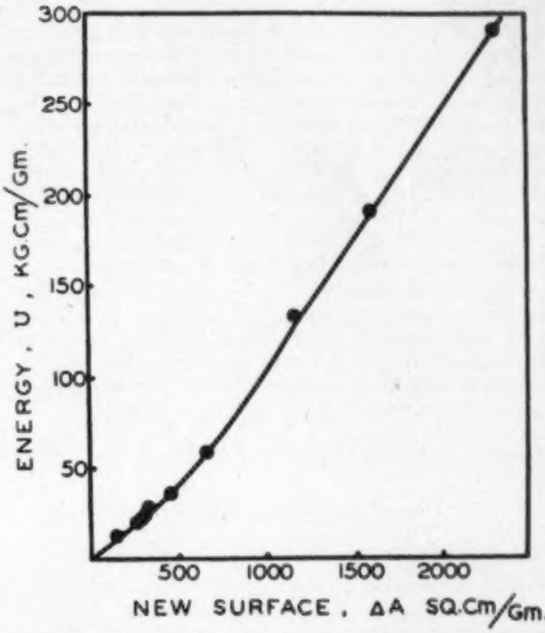

Fig. 12—Energy vs change in surface area, after Piret.

This trend of increased hardness is another indication of the size effect, Eq. 12. The diamond indentor is geometrically similar for all depths of penetration; thus if the material were homogeneous in its properties, the same hardness value should be obtained for all impression depths and hence for all loads. However, the plastic flow stress of the marble is seen to increase as the volume of the material deformed is decreased, indicating that the material is not homogeneous with regard to size.

In all tests reported here, the critical particle thickness varied from 25 to 50 micro in. X-ray diffraction and electron microscope studies have revealed the mean spacings of the inhomogeneities in metals to be about 1 micron. Rosenthal and Kaufman[7] have recently investigated the size of a perfect crystallite of Yule marble. The marble was first plastically deformed from 15 to 25 pct while a hydrostatic pressure of 10,000 atmospheres was acting to prevent rupture. The deformed specimen was then crushed and sized, and the X-ray line broadening of specimens of different particle size determined. Line broadening was found to disappear in specimens below 1 micron in size. These experiments show that the line broadening is due to residual stresses between crystallites. They also establish the size of the particle within which no plastic deformation occurred at 1 micron. In the grinding data given in Fig. 7 the critical depth of cut below which theoretical values of strength were achieved was found to be about 50 μ in or about 1.25 micron. This value is seen to be in good agreement with the results of Rosenthal and Kaufman for marble.

The close similarity between machine grinding of metals and the comminution of materials normally considered brittle is shown by their tendency to follow the same energy laws. On closer examination the difference in the ductility of the materials normally ground in these two processes is not found to be real. The usually brittle substances that are normally crushed are really quite ductile in the size range in which comminution occurs. A schematic

diagram of a particle being crushed in a ball mill is shown in Fig. 11. Since the particle will be small compared with the size of the balls, it can readily be forced to shear when compressed between the balls. If the particle is small enough so that it contains no inhomogeneities it should be expected to shear along the planes indicated and to exhibit its theoretical strength. The energy relation for this type of action will follow the law of Kick. If an inhomogeneity is present in the particle as at A, rupture will occur on a plane passing through this point and a stress level below the theoretical value will be obtained.

Piret and his associates have reported[4] test results in which crystalline quartz was crushed in a carefully controlled drop test. The resulting area was determined by the gas adsorption method. Representative values of surface area are shown plotted against the corresponding crushing energy in Fig. 12. According to Rittinger's law, Eq. 2, this should be a straight line. However, the slope of the curve

$$\frac{dV}{d(\Delta A)} = K_2$$

is seen to increase as the particle size is decreased, i.e., as the area, ΔA, generated increases. This observed increase in K_2 with decreased particle size is in qualitative agreement with Table II for the minerals and the curves of Fig. 6 for the metal specimen. This is further evidenced in support of the observation that Rittinger's law does not provide a large enough size effect. However, the results of Piret et al shown in Fig. 12 somewhat mask the actual deviation from Rittinger's law, in as much as the values plotted represent the result of several blows in a drop test. The values of energy plotted are thus the mean values for the range of particles produced in the several blows. Owing to the very strong size effect that has been shown to exist in crushing, a truer energy picture is obtained when the energies considered refer to a single particle size rather than to a range of particle sizes.

Concluding Remarks

The importance of the particle size upon the energy required to crush a material has not been recognized in previous comminution studies. Instead, the physically unimportant surface area has been adopted as the variable against which to correlate energy consumption. Furthermore, comminution experiments in the past have been concerned with energies associated with a range of particle sizes rather than with particles of one size. This has led to considerable controversy concerning the relationship between energy and surface area.

In the machine grinding operation, the energy consumed in particle formation can be studied under conditions that provide particles of essentially constant size. When data thus obtained is analyzed it is found that Kick's law holds for very fine grinding, i.e., for particles less than about 1 micron. On the other hand, Rittinger's law is found to be in agreement with experimental data only when the range of particle size involved is relatively small. When the data considered covers an appreciable range of particle size, Rittinger's law predicts too small a size effect. Kick's law naturally arises from the concept of a theoretical strength in the region of extremely fine particle size, and from consideration of properties of the material in bulk when in the region of extremely coarse particle size. However, although Rittinger's law qualitatively holds quite well over limited size ranges, there is no apparent physical basis for Rittinger's prediction that is in agreement with current concepts.

In this paper it has been indicated that appreciable plastic deformation is associated with the grinding of materials ordinarily considered brittle. For this reason there is little difference between machine grinding and other comminution processes such as ball milling. It would appear that the machine grinding technique described here offers a precise means for evaluating the grinding characteristics of various materials.

Acknowledgments

This study of the comminution process was supported by grant No. 202 of the National Science Foundation and is an outgrowth of an investigation of the grinding of metals sponsored by the Carborundum Co. The authors wish to acknowledge the help of Professor W. H. Dennen of the Geology Department in providing the mineral specimens and much useful information. Mr. W. E. Littman of the Metallurgy Department was kind enough to help in the preparation of the indentation photomicrographs, and Professor H. Rush Spedden, formerly of the Metallurgy Department, provided important literature and information. It should be mentioned that the idea that a connection might exist between machine grinding and ball milling was originally expressed by Professor T. K. Sherwood of the Chemical Engineering Department in a conversation in the authors' laboratory two years ago.

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Mining Branch Features Heavy 1954 Program

Plans for Future Meetings on Tap

Members of the Mining Branch of AIME can look forward to a decidedly active year, with meetings of the Industrial Mineral Div., Minerals Beneficiation Div., and the Rocky Mountain Region Industrial Minerals Conference already in the works.

Fall meeting of the Industrial Minerals Div. is scheduled for Lake Placid, October 5 to 9. The Adirondack Section will be host to the gathering, which will encompass what is expected to be an outstanding technical program. October is a particularly fine time of year to visit Lake Placid. Headquarters for the meeting will be the Whiteface Inn.

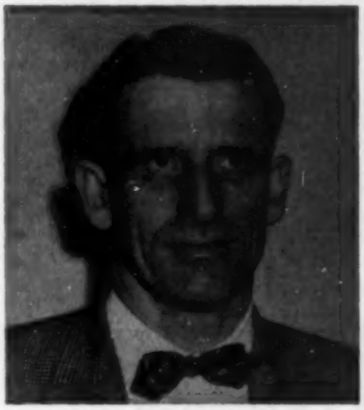
The Rocky Mountain Region Industrial Minerals Conference, slated for Salt Lake City, will hold registration October 28, with sessions on October 29 and 30. W. F. Rappold, Vice Chairman, Industrial Minerals Div., Rocky Mountain Region, feels that papers will transcend a wide field of interest among members of the AIME living in these states.

Minerals Beneficiation Div. has completed arrangements to hold its Fall Meeting in San Francisco, September 24. The meeting will follow the 4-day 1954 Mining Show of the American Mining Congress.

Once again, the Coal Div. of the AIME joins the American Society of Mechanical Engineers in the Annual Fuels Conference. This year, the meeting will be held October 28 and 29 at the William Penn Hotel, Pittsburgh.

While the Mining, Geology, and Geophysics Div. is not planning a national Divisional fall meeting, MGGD is programing several regional meetings in cooperation with mining area local sections. The Division intends to stimulate group activities beyond the February Annual Meetings by better distribution of information regarding local and regional gatherings.

Program Set for Northwest Conference



L. C. RICHARDS
Ind. Min. Div. Chairman

Francis X. Cappa, smelting div. mechanical engineer at the Vancouver, Wash., works of the Aluminum Co. of America, is serving as General Chairman of the AIME Pacific Northwest Metals and Minerals Conference to be held Apr. 29 to May 1, 1954, at the Multnomah Hotel in Portland, Ore.

Leslie C. Richards, mining engineer of Portland, Ore., is the Pacific Northwestern Chairman for the Industrial Minerals Div. Ind. Min. Div. participation in the Pacific Conference will consist of three sessions. One of these will feature a symposium on ground water resources with A. M. Piper, staff scientist of the USGS, as moderator.

Two other sessions will be devoted to nine papers on the occurrence, mining, processing, transportation, and marketing of the West's nonmetallics. Chairmen for these are: T. M. Robins, Major General U. S. Army, Ret., president, Raw Materials Survey, Inc. Portland, Ore.; V. E. Scheid, dean, Mackay School of Mines, University of Nevada; H. W. Marsh, secretary, Idaho Mining Assn.; and G. H. Waterman, president, Manufactures Minerals Co., Seattle.

Closer Cooperation Between CIM And AIME Visualized by Committee

Five concrete suggestions for evolving closer relations between the AIME and the Canadian Institute of Mining and Metallurgy have been made by Sherwin F. Kelly, Chairman of the AIME Committee on Cooperation with the CIM.

Suggestions submitted to the other 23 committee members, in a letter which reviewed progress resulting from inquiries and discussions, were: (1) that Institute publications give the general nature of the next annual meeting of the sister society; (2) that the Institutes make names and addresses of officers of border sections available to each other, and that local section mailing lists include members, or at least secretaries of the nearest local section across the border, thus promoting attendance at meetings of border sections; (3) that a mutual assistance program for annual general meetings be initiated by inviting members of the sister Institute to present papers at appropriate technical sessions; (4) that interchange of CIM and AIME papers be further developed by including in the membership of each Divisional technical committee the name of the Chairman or other representative of the corresponding committee of the other Institute; and (5) that joint meetings of similar technical Divisions be fostered within the Institutes.

Mr. Kelly pointed to the successful joint meeting at the Keltic Lodge, Nova Scotia, last September, as an example of cooperation between societies.

William E. Wrather Honored at Waldorf-Astoria Dinner

William E. Wrather, director of the U. S. Geological Survey, Washington, D. C., received the John Fritz Medal at a dinner in his honor at the Waldorf-Astoria, New York. The medal is given jointly by the four founder societies, American Institute of Mining and Metallurgical Engineers, of which Mr. Wrather is a Member; American Society of Civil Engineers; American Society of Mechanical Engineers; and American Institute of Electrical Engineers. Seated from left to right, outer rim: John Keshishian, T. B. Counselman, Erle V. Daveler, Louis S. Cates, John M. Lovejoy, Felix E. Wormser, Fred M. Nelson, John Suman, William E. Wrather, Andrew Fletcher, D. H. McLaughlin, Edward H. Robie, J. Ed. Warren, J. Terry Duce, Michael L. Haider, James Boyd, Reginald Burbank, and O. B. J. Fraser. Seated on the inside rim, from left to right, are: Willis Mc. G. Peirce, Russell B. Caples, A. B. Kinzel, C. M. Cooley, Clyde Williams, E. O. Kirkendall, H. DeWitt Smith, Thomas B. Nolan, Ernest Hartford, M. D. Cooper, and James L. Head.

Mr. Wrather was chosen to receive the medal by a 16-man board composed of representatives of the four societies. The citation accompanying the medal reads: "A geologist of world wide experience and fame; an outstanding scientist and historian; a wise leader distinguished for his service to the nation."

He was one of the organizers of the 16th International Geological Congress in Washington. He is also noted as a gatherer of historical data whose collection of maps has been consulted by the Library of Congress. He once served as president of the Texas Historical Society. Mr. Wrather's travels in search of geological information have taken him to the Balkans, Africa, South America, and Asia.

Around the Sections

• Harold A. Krueger, manager, St. Louis Smelting & Refining Div., National Lead Co., pointed out that a new chemical process now being readied for operation will improve recovery of nickel and cobalt by at least 100 pct over any system used before. The statement was made at a recent meeting of the **St. Louis Section,** during which Mr. Krueger spoke on *Treatment of Fredericktown Complex Ores.*

• First meeting of the **Washington, D. C., Section** under its new officers heard Andrew Fletcher, AIME President, speak on the award of the John Fritz Medal to William Wrather, who is director of the U. S. Geological Survey. Samuel G. Lasky is the new Chairman of the group, with Joe McBrian and Edwin J. Lintner, as Vice Chairmen. Robert D. Thomson is Secretary-Treasurer, and C. J. Williamson and W. E. Wright have been elected Members of the Executive Committee.

• AIME President Andrew Fletcher, and E. O. Kirkendall, Assistant Secretary, spoke before the **Boston Section** on Institute Matters. At another meeting Harold B. Ewoldt spoke to section members on the White Pine project and Bruce Chalmers, Harvard University, addressed the group on *Metallurgy in the U. S., Canada, and the United Kingdom.*

• A color motion picture, *Man with a Thousand Hands,* produced by International Harvester Co., was featured at a recent meeting of the **Alaska Section** at College, Alaska. The movie deals with the engineering, construction, and transportation involved in the Kitimat-Keemano project.

• Gordon S. de Villiers, mine manager, Hartebeestfontein Gold Mine, Orange Free State, Union of South Africa, discussed mining operations in his home region at a recent meeting of the **Carlsbad Potash Section,** Carlsbad, N. M. Future meetings of the section will be held on the third Wednesday of each month at the Riverside Country Club, Carlsbad.

• Durand A. Hall spoke at the first 1954 meeting of the **San Francisco Section** on the status of the domestic chrome industry. Mr. Hall and George I. Barnett operate the Castro chrome mine in San Luis Obispo County, Calif. A discussion of the relative merits of the Government stockpiling program followed.

• Highlighting the meeting of the **Southern Sierra Subsection,** in Lone Pine, Calif., was an address by Tom Edson, vice president of American Potash & Chemical Corp. He spoke on the processing of phosphate rock. Merle Otto presided at the meeting.

• ***Ultrasonic Inspection,* a 10 min sound-color motion picture describes development, theory, operation, and application of Sperry Ultrasonic Reflectoscope for nondestructive testing of metals and other materials. It can be obtained from Sperry Products Inc., Danbury, Conn.**

• **Air Reduction Sales Co. has produced a 16 mm color motion picture about atmospheric gases, covering uses in industry, and various ways they appear in daily life. The film contains some dramatic photography showing oxygen in steel refining and in hospital oxygen tents; welding, helium for lighter-than-air craft; in nylon manufacture. The film, *Whatever We Do,* can be obtained at district offices, or by contacting Air Reduction Sales Co., 60 E. 42nd St., New York.**

ECPD Lists Accredited Mineral Engineering Colleges

As a service to its readers, MINING ENGINEERING is publishing the mineral industry portion of the 1953 list of "Accredited Undergraduate Engineering Curricula in the United States." This list, revised annually, is issued by the Education Committee of the Engineers' Council for Professional Development, and covers all phases of engineering education. The AIME is actively represented on ECPD, which was jointly organized by the Founder Societies and various other groups interested in the professional recognition of engineers.

Engineers' Council for Professional Development has as its expressed objective the enhancement of the status of the engineering profession. To this end ECPD has a program dealing with selection, guidance, training, and recognition of the members of the profession. The program is carried out under the direction of seven committees, one of which, the Education Committee, has as part of its broad purpose "to formulate criteria for colleges of engineering which will insure to their graduates a sound educational background for practicing the engineering profession."

At the time the committee was constituted there was danger of a multiplicity of accredited lists and accrediting agencies. No one list was accepted on a national basis, nor could there be any assurance of a country-wide uniformity of high standards. One of the major purposes of ECPD was the amelioration of these unsatisfactory conditions regarding accrediting.

ECPD is merely authorized by its constituent organizations to publish a list of accredited engineering curricula for use as desired by those agencies which require such a list. It has no authority to impose any restrictions or standardizations upon engineering colleges, nor does it desire to do so. On the contrary it aims to preserve the independence of action of individual institutions and to promote thereby the general advancement of engineering education. *(Date following school refers to year of initial accrediting.)*

GEOLOGY

Arizona, University of (1950)
Colorado School of Mines
Idaho, University of (1950)
Michigan College of Mining and Technology (1951)
Minnesota, University of (1950)
Montana School of Mines
Oklahoma, University of (1953)
Pittsburgh, University of (1950)
Princeton University (1949)
Saint Louis University (1951)
South Dakota School of Mines (1950)
Texas, A. and M. College of (1949)
Utah, University of (1952)
Washington University (1948)

GEOPHYSICS

Colorado School of Mines (1953)
Saint Louis University (1951)

METALLURGICAL ENGINEERING

Alabama, University of (1949)
Arizona, University of (1950)
California, University of (Berkeley)
Carnegie Institute of Technology
Case Institute of Technology
Cincinnati, University of (1948)
Colorado School of Mines
Columbia University
Cornell University (1951)
Drexel Institute of Technology (1953)
Fenn College (1948)
Harvard University (Physical Metallurgy)
Idaho, University of (Metallurgy) (1938)
Illinois Institute of Technology (1949)
Illinois, University of
Kansas, University of (1953)
Kentucky, University of
Lafayette College
Lehigh University
Massachusetts Institute of Technology (Metallurgy)
Michigan College of Mining and Technology
Michigan, University of
Minnesota, University of
Missouri School of Mines and Metallurgy
Montana School of Mines
Notre Dame, University of (Metallurgy) (1942)
Ohio State University
Pennsylvania State University (Metallurgy) (1938)
Pennsylvania, University of (1949)
Pittsburgh, University of
Purdue University (1941)
Rensselaer Polytechnic Institute (1938)
South Dakota School of Mines
Stanford University (1952)
Utah, University of
Virginia Polytechnic Institute (1948)
Washington, State College of
Washington, University of
Wayne University (1950)
Wisconsin, University of
Yale University (Metallurgy)

MINING ENGINEERING

Alabama, University of
Alaska, University of (includes Geological 5-yr. option) (1941)
Arizona, University of
California, University of (Berkeley)
Colorado School of Mines
Columbia University
Idaho, University of (1938)
Illinois, University of
Kansas, University of (1953)
Kentucky, University of
Lafayette College
Lehigh University
Michigan College of Mining and Technology
Minnesota, University of
Missouri School of Mines and Metallurgy (Mine; includes Petroleum option (1941); Mining Geology option (1950)
Montana School of Mines
Nevada, University of
North Dakota, University of
Ohio State University (Mine)
Pennsylvania State University (1938)
Pittsburgh, University of
South Dakota School of Mines
Stanford University (1952)
Texas Western College (formerly Texas College of Mines and Metallurgy) (1947) (includes option in Mining Geology and Metallurgy)
Utah, University of
Virginia Polytechnic Institute (1948)
Washington, State College of
Washington, University of
West Virginia University
Wisconsin, University of

Personals

Keith O'Donnell has been appointed mine superintendent of Frontino Gold Mines Ltd. in Colombia.

R. K. Barcus is chief engineer, Mountain Copper Co. Ltd., Matheson, Calif.

John R. Nichols has been named sales agent in the explosives sales div. for Atlas Powder Co. in Spokane, Wash. Mr. Nichols was formerly a Seattle district salesman. He joined Atlas in 1945 when he returned to the U. S. after being held prisoner by the Japanese for more than 2 years. He was captured while mining and quarrying in the Philippines.

PHILIP D. WILSON

Philip D. Wilson has announced an arrangement with Lehman Bros. and the Lehman Corp., New York, whereby a portion of his time will be available for consultation and for the examination and evaluation of mines and of mining securities for other clients.

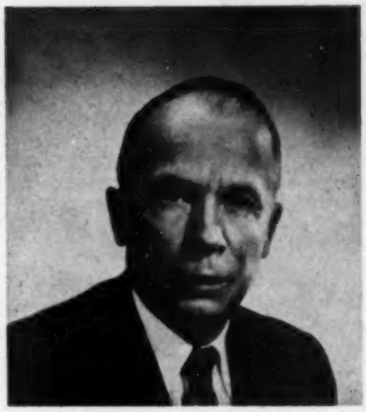
David F. Lleras is with Minas de Matahambre, Pinar del Rio, Cuba.

Malcolm Hill, formerly with Staff Quarters, M.I.M. Ltd., Mount Isa, Queensland, Australia, has been appointed lecturer in oredressing, Dept. of Mining, Metallurgical and Chemical Engineering, University of Adelaide, Adelaide, Australia.

John C. Keenan who was with Resurrection Mining Co., Leadville, Colo., is now with Idarado Mining Co. in Ouray, Colo.

Neal M. Muir is engaged in examination work for the mining div. of the U. S. Bureau of Mines, with headquarters in Spokane.

H. W. Straley, III, of Princeton, W. Va., is currently engaged in geophysical work on iron deposits in the State of Georgia.



HOWARD M. ZOERB

Jack B. Bond has been appointed assistant general manager and **Howard M. Zoerb** has been appointed divisional consulting engineer of the crusher, screen & process machinery div., Nordberg Mfg. Co., Milwaukee. Before joining Nordberg in 1944 as sales engineer, Mr. Bond served in an engineering capacity for the A. O. Smith Corp. Mr. Zoerb has been with Nordberg for over 30 years and is acquainted with the field problems of mining communities in North, Central, and South America.

Clyde E. Weed has been elected president and **Edward S. McGlone** and **Albert Mendelsohn** have been elected vice presidents, respectively, of Greene Cananea Copper Co., a subsidiary of Anaconda Copper Mining Co. Mr. Weed and Mr. McGlone and Mr. Mendelsohn are also directors of Greene Cananea Copper Co. Clyde E. Weed is vice president in charge of operations and a director of Anaconda Copper Mining Co. Edward S. McGlone is executive vice president and also a director of Anaconda Copper Mining Co. Albert Mendelsohn is also president and general manager of Greene Cananea's subsidiary, the Cananea Consolidated Copper Co. S. A., located in Cananea, Sonora, Mexico. He has held these positions for the past 15 years.

D. D. Morris has been appointed administrative assistant, Consolidated Mining & Smelting Co., Trail, B. C. Succeeding Mr. Morris as manager of the research and development div. is **A. D. Turnbull,** who was assistant manager of the metallurgical div. **T. H. Weldon,** superintendent of the zinc dept., has been appointed general superintendent of the metallurgical div. Mr. Morris joined COMINCO in 1928, Mr. Turnbull in 1925, and Mr. Weldon in 1923.

Priestley Toulmin, III, is on leave from his position as geologist with the U. S. Geological Survey. He is a graduate student at Harvard University.

F. Blondel of Paris has been elected president of the Académie des Sciences Coloniales and also vice president of the Société des Ingénieurs Civils de France.

Hans Pick, who was with Cia. Minera Aguilar, Tres Cruces, Argentina, is with The Eimco Corp., Salt Lake City.

Henry G. Schuring is now with Atlas Consolidated Mining & Development Co. as mill superintendent of the new 4000-ton capacity copper concentrator at Toledo, Cebu, Philippines. Mr. Schuring was mill superintendent of Neptune Gold Mining Co., Bonanza, Nicaragua.

Gale A. Hansen has accepted the position of mine superintendent for the Silver Buckle Mining Co., Wallace, Idaho.

Robert A. Lubker, formerly associate manager, has been appointed manager of the metals research dept. at Armour Research Foundation, Illinois Institute of Technology, Chicago. Mr. Lubker graduated from the University of Washington in 1942 and received his master's degree at Carnegie Institute of Technology in 1946. At 33 years old, he now heads one of ARF's largest research departments, replacing **Max Hansen** who resigned to devote full time to revising and translating from the German his textbook on phase diagrams.

LEWIS L. HUELSDONK

Lewis L. Huelsdonk, secretary-treasurer and general manager of Best Mines Co. Inc., Downieville, Calif., has been appointed to the California State Mining Board to fill the vacancy created by the term expiration of **W. Wallace Mein,** president of Calaveras Cement Co., San Francisco.

Russell C. Nelson, metallurgical engineer, is with Sylvania Electric Products Inc., tungsten & chemical div., Towanoa, Pa.

Robert L. Swain has been appointed chief mine engineer, Braden Copper Co., Rancagua, Chile.

Charles D. Michaelson is vice president of Braden Copper Co., a subsidiary of Kennecott Copper Corp. Mr. Michaelson will be the chief executive of the company in Chile with offices in Santiago. He became associated with Braden Copper Co. in Chile in 1948 as general superintendent and in 1952 was advanced to general manager. Mr. Michaelson is also a director of the company.

William C. Chase, executive assistant and general mining consultant, Alabama By-Products Corp., Birmingham, has retired.

E. R. Price, manager, coal properties, Inland Steel Co., Wheelwright, Ky., is retiring after 24 years of service. Succeeding Mr. Price as manager is **John T. Parker**. Mr. Parker joined Inland in 1926 as an engineer at the company's coal mines at Indianola, Pa. Four years later he was transferred to Wheelwright where he has served as mine engineer, mine superintendent, and general superintendent.

C. D. Rubert is resident engineer, Pennsylvania Turnpike Commission, Lansdale, Pa.

J. D. Warfel was recently on the Associated Press Wirefoto machines. Just before commencement exercises at Penn. State University he was photographed with his four children, all of whom have been born since he enrolled to study mining engineering. The picture had nationwide distribution.

L. W. Hartman has been visiting in New York from South Africa. Mr. Hartman, who was formerly a geologist with Climax Molybdenum Co., Climax, Colo., recently completed his doctcrate in geology at the University of South Africa, Johannesburg. Mr. Hartman is returning to work in South Africa.

William J. McCaughey, chairman of the dept. of mineralogy, Ohio State University, has been named the 1954 recipient of the Albert Victor Bleininger award. This award for "distinguished achievement in the field of ceramics" is presented annually by the Pittsburgh Section of the American Ceramic Society.

Warren R. Philbrook, in charge of industrial relations for Food Machinery & Chemical Corp.'s chemical divisions, New York, has filled the vacancy created by the appointment of FMC's industrial relations director **Albert C. Beeson** to the National Labor Relations Board in Washington, D. C. Mr. Philbrook will hold this position during Mr. Beeson's leave of absence until December 1954.

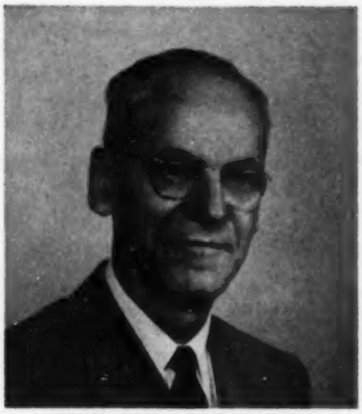
Walter J. Williams, who has been deputy general manager of the Atomic Energy Commission since 1951 and its virtual chief since 1946, has resigned to enter private industry. Speaking of Mr. Williams' resignation, AEC general manager **K. D. Nichols** said that "if there was an irreplaceable man in atomic energy, Walt comes closest to being that man." Mr. Williams is to be vice president of the Taconite Contracting Corp., a subsidiary of the Erie Mining Co. He will direct development of the taconite operations north of Duluth.

Karl R. Fleischman has resigned as Inspector of Mines, Mines Dept., Herberton, North Queensland, Australia, to accept the position of Inspector of Mines, Suva, Fiji.

F. A. McGonigle, general manager of Manganese Inc., has been named vice president of the firm. He will continue as general manager of the company's plant in Henderson, Nev.

Raymond E. Zimmerman has resigned as chief preparation engineer, U. S. Steel Corp., to accept the position of vice president, the Paul Weir Co., Chicago.

Thomas Edson is assistant vice president, research and development, American Potash & Chemical Corp., Los Angeles.



A. W. FAHRENWALD

A. W. Fahrenwald, dean of the School of Mines, University of Idaho, for 21 years, will retire September 1. He will do consulting work and establish an office with General Machinery Co., Spokane, manufacturers of the ore and aggregate processing equipment developed by him. Mr. Fahrenwald first came to the University of Idaho in 1919 as an ore dressing engineer with the U. S. Bureau of Mines, and joined the university staff in 1929 as professor of metallurgy.

Thorndike Saville, dean of New York University's College of Engineering, has been elected president of Engineers' Joint Council.

William T. Mahood, formerly district manager Atlas Powder Co.'s San Francisco explosives sales district, has been recalled to the Wilmington general office for a post in the contractors section of the explosives sales div. **Eugene F. Daily,** who was a special field representative, succeeds Mr. Mahood as district manager of the San Francisco office. **Harry E. Urben,** formerly with the Seattle office, is now assistant manager of the San Francisco district.

William E. Smith is now assistant sales manager of The Dorr Co., Stamford, Conn. Replacing him as manager of North American Industrial sales in Stamford is **Glenn O. Wilson,** previously manager of the Western Industrial Div. in Denver. This position is now held by **William T. Marston.** Mr. Marston, a graduate of the University of Toronto in mining and metallurgy, joined Dorr in 1949. During World War II, he was a pilot officer in the Royal Canadian Air Force in Europe. Mr. Smith has been with the company since 1924, shortly after graduating from the University of Colorado. Mr. Wilson, a University of Denver graduate, joined Dorr in 1928.

J. B. Stapler, a vice president of Marsman & Co. Inc., Manila, and manager of the Mines Div. for many years, was recently appointed consulting engineer for the company and will be leaving the Philippines soon for the U. S. on vacation.

William Morrow, Jones & Laughlin Steel Corp., Pittsburgh, has been transferred to the firm's Minnesota Ore div. as director of training and employment on the Mesabi Range, with headquarters at Virginia, Minn.

J. Joseph Kelleher has been made sales manager of the explosives dept. of Hercules Powder Co., Wilmington, Del. Mr. Kelleher joined the explosives dept. in 1929.

V. A. Zandon has been appointed plant superintendent, Southwest Potash Corp., Carlsbad, N. M.

Raymond C. Troxell is employed by the Aluminum Co. of America in the East St. Louis, Ill., aluminum research laboratories as a research chemist. Mr. Troxell received an M.S. degree in mineral preparation from Pennsylvania State University in January.

J. M. Hawkins, New York, comptroller for Phelps Dodge Corp., has recently been elected a vice president of the firm. He will continue as comptroller in addition to his new duties.

Frank R. Milliken, vice president of Kennecott Copper Corp., has been appointed chairman of the national program committee for the American Mining Congress' Metal and Nonmetallic Mining Convention and Exposition to be held in San Francisco, September 20 to 24.

Paul M. Johnson of Lakewood, Ohio, has been named sales engineer for General Refractories Co. in the Cleveland area. Mr. Johnson was formerly an open hearth consultant with the Iron & Steel Div., Arthur G. McKee & Co.

I. E. Janelid, professor, Royal Institute of Technology, Stockholm, Sweden, who presented a paper on Swedish drilling trends at the AIME Annual Meeting, is now visiting mines and universities in the U. S.

Edward F. McCrossin was recently elected a director of Yuba Consolidated Gold Fields. Mr. McCrossin is a mining engineer and executive head of McCrossin & Co., New York consulting engineers.

Felix Wormser, assistant secretary for mineral resources of the Dept. of the Interior, has been named by the Office of Defense Mobilization as one of the three members of a titanium advisory committee.

James E. Reynolds is with Battelle Memorial Institute, Columbus, Ohio.

Obituaries

William R. Allen (Member 1947) died Oct. 31, 1953. He was chairman of the board and consulting engineer for Elkhorn-Beverhead Mines Co., Butte, Mont. He was born in 1871 in French Gulett, Mont., and received his early education in Montana. From 1902 to 1906 he was superintendent of French Gulch Dredging Co. He later was president of Allen Gold Mining Co., Boston Montana Corp., and mines manager for Mining Associates Inc.

George S. Baton (Member 1919) died Dec. 28, 1953 after a brief illness. A prominent figure in mining engineering throughout the Pennsylvania-Ohio-West Virginia coal fields, Mr. Baton was a partner in George S. Baton & Co., Pittsburgh. This firm directed over 40 mining installations and plants across the country. Mr. Baton was also chairman of the board of the Greensburgh-Connellsville Coal & Coke Co., which operated several mining developments in the Tri-State Coal Area. He was born in Philadelphia in 1869 and after graduating from Lehigh University with a B.S. in mining, became the engineer in charge of the Monongah Coal & Coke Co. in West Virginia. He was later division engineer for H. C. Frick Coke Co., Scottdale, Pa. After operating coal mines in Fayette County, Pa., he became a consulting mining engineer. Mr. Baton was past president and treasurer of the Engineers Society of Western Pennsylvania and belonged to ASCE, Tau Beta Pi, and various Masonic organizations.

Serge Bogroff (Member 1952) died Sept. 24, 1953. He was a mining engineer, consultant for Rothchilds Frères in Paris, a director of Moisant Laurent Savey, Société Nationale de Construction, and other companies. Mr. Bogroff studied at the Ecole Nationale Supérieure des Mines and the Faculté de Droit. A lieutenant in the French Army, he was a German prisoner of war from 1940 to 1945.

John L. Dynan (Member 1915) died Nov. 24, 1953, in Tonopah, Nev., where he was for many years superintendent of the Tonopah Extension Mine Co. Mr. Dynan was born in Norwich, N. Y., in 1890 and graduated with an E.M. from Lehigh University in 1909. His early mining experience was gained in Idaho, Utah, and British Columbia. Mr. Dynan was at various times general superintendent of the Belmont-MacNeill mine, Palo Verde, Ariz., superintendent of Tonopah Belmont Development Co., and superintendent of the Mt. Gaines Mining Co., Hornitos, Calif.

Appreciation of
Walter Irving Garms
by W. Sprott Boyd

Walter Irving Garms (Member 1941) died at his home in **San Mateo,** Calif., on Dec. 29, 1953. Born on Sept. 7, 1886, he spent his boyhood in San Francisco and graduated from the University of California in 1910 with a B.S. in mining. On Sept. 20, 1922, he married Margaret Dehm, daughter of Mr. and Mrs. J. F. Dehm of San Diego, Calif., who survives him, also a son, Walter I Garms, Jr., and two grandchildren; and a daughter, Mrs. Mary Alice Ramsden.

Almost immediately following graduation he entered the employ of Ray Consolidated Copper Co., which later became the Ray Mines Div. of Kennecott Copper Corp. After a few months at the mine, he transferred in February 1911 to the milling department at Hayden, Ariz., where the first concentrator was nearing completion, and remained there until his retirement on Oct. 1, 1951. He rapidly worked his way up, becoming general mill foreman in 1915. In 1917 he left for military service in World War I; returning in 1919 he was promoted to mill superintendent in 1926, and was made assistant general manager of Ray Mines Div. in 1943, in which capacity he acted until his retirement.

He loved his work and was interested in every phase of milling operation and he had the inquisitive and reflective type of mind that successful research requires. He promoted the first use of the higher alcohol xanthates in the treatment of copper ores containing important quantities of their content in semi-oxidized form. The value of these compounds as flotation reagents was discovered by J. L. Stevens, then research chemist and now mill superintendent at Hayden. In later years, it was the problems pertaining to fine crushing and fine grinding that occupied most of Walter Garms' attention and his investigations covered a number of years, during which he engaged in many discussions at technical meetings, carried on a wide correspondence with others in the same field, was the author of several papers, and developed what is known as the Garms Ratio which has found acceptance in the literature on milling.

During his years at Hayden so many changes in milling methods and equipment were developed that practically nothing of the original concentrator exists today. Since 1926, most of the alterations were in accord with his suggestions which came to be more and more depended upon as the correctness of his conclusions was demonstrated by results and the last and almost complete rebuilding of the Hayden concentrator, which was finished under his supervision just before his retirement, was wholly in accord with his recommendations. The capacity of the plant has been increased to 15,000 tons a day to accommodate the larger tonnages resulting from the change to pit methods at the mine.

He was an able administrator. He knew every man on the payroll, respected them as individuals, and gave full recognition to their efforts in the common cause and they, in turn, respected him. At the Hayden mill absenteeism was at a minimum and the labor turn-over almost negligible except when caused by economic conditions and reduction of operations. During his 40 years at Hayden no man ever lost a shift nor did a wheel ever cease to turn because of a labor controversy, which happy situation still continues as of today. A harmonious labor relationship in any industry depends as much on the personality of the man on the job who represents the company as on the corporation's labor policy.

I, too, went to Arizona in those early days. In fact, I arrived just a few days before Walter and this early acquaintanceship developed into a friendship that grew strong and constant throughout the years. It was a privilege and a delight to know Walter really well for he was a truly fine person, clean and wholesome, modest, and almost shy as to his own accomplishments, but ever ready to give praise and credit to others. He was conscious and mindful of his responsibilities, both on the job and off, and loyal and true

to his country, his company, his community, his family, and to himself. His was a useful life.

Pierre E. Henry (Member 1930) died Dec. 3, 1953 as a result of an auto accident in France. He was well known in iron ore mining and processing circles in the U. S. as well as Europe. Mr. Henry was consulting engineer for the Cie de Mokta-el-Hadid in Paris. He was born in Paris in 1889 and after attending the Ecole des Mines de St. Etienne, worked as a mining engineer with the Société Industrielle et Métallurgique du Caucase in Russia. Following service with the French army in World War I, he joined the staff of MM. Schneider et Cie, Le Creusot, France, and he prospected for this company in China and French Indo-China. His later positions took him to Poland, Lithuania, the Saar Basin, and then to Algiers where he was in charge of various mines.

Franklin D. Howell (Member 1935) died Sept. 10, 1953. He was a Los Angeles civil and mining engineer, a consultant on mine examinations, hydro-electric and electric railway plants, and mining and milling methods. Mr. Howell was born in Pennsylvania in 1867 and was educated at the state university. After working for 8 years as an assistant engineer for Pennsylvania R.R., surveying and constructing branch lines in the anthracite and bituminous coal fields in Pennsylvania and West Virginia, he became chief engineer for the Cannelton Coal Co., West Virginia. He later worked for J. W. Hoffman & Co., Philadelphia, Engineering Contract Co. and T. A. Gillespie Co., New York. In 1902 Mr. Howell went to Los Angeles to be assistant engineer for Pacific Electric Railway.

Sherwin P. Lowe (Member 1943) died Oct. 18, 1953. He was vice president of the Tin Processing Corp., Texas City, Texas. Born in Denver in 1894, he gained his early professional experience in cyanide mills in Nevada and Utah. Mr. Lowe was for many years mill and research superintendent for Hudson Bay Mining & Smelting Co. Ltd., Flin Flon, Manitoba. An active member of the Canadian Institute of Mining & Mettallurgy, he had served as vice president for a year and a member of the executive council for 2 years. Two articles by him cn the Hudson Bay Mining & Smelting Co. were published by the CIM.

Robert W. Michael (Member 1937) died Dec. 22, 1953. He was a New York mining engineer and for many years chief of the technical section, Cia. de Diamantes de Angola, Dundo, Angola, Portuguese West Africa. He gained early experience as an assistant in an assay office in Los Angeles, and was later superintendent of the Wolframite mine in California. From 1917 to 1920 Mr. Michael was superintendent, Illinois Arizona Copper Co., Parker, Ariz. He first went to work for Cia. de Diamantes de Angola in 1921. He was also associated with the Cia. Anomina Minera Gran Sabana, Gran Sabana Syndicate, as field manager.

Arthur T. Ward (Member 1916) died Dec. 15, 1953. He was a mining and metallurgical engineer and president of Colloid Equipment Co. in New York. Mr. Ward was born in 1890 and studied electrometallurgy at Lehigh University. He then worked as chemist with the U. S. Metals Refining Co. in Chrome, N. J. In 1914 he went to Chile with the Braden Copper Co. as a research engineer. He was later with Union Carbide Sales Co. in Havana, Cuba, for several years. Mr. Ward was a member of the Engineers Club and the Mining Club in New York.

Pope Yeatman (Member 1883) died Dec. 5, 1953 in Chestnut Hill, Pa. An internationally known mining engineer, he was awarded the William Lawrence Saunders Gold Medal in 1934, "in recognition of his distinguished achievements as a mining engineer, for his vision in recognizing the value of mineral deposits and his ability in successfully developing them; and for his outstanding executive ability in the management of extensive operations." Mr. Yeatman was born in St. Louis in 1861, and received his E.M. from Washington University in 1883. His first experience in mining was gained in Mexico, Missouri, and the Southwest. In 1895 he went to South Africa where he managed several Rand gold mines. Returning to the U. S. in 1904 as chief engineer for the Guggenheim mining interests, he helped develop the large Chilian copper mines and also investigated Guggenheim interests in Alaska and China. During World War I Mr. Yeatman served as a dollar-a-year man under Bernard Baruch. He was director of the nonferrous dept. of the War Industries Board and as a member of that board received the Distinguished Service Medal. After World War I Mr. Yeatman made a study of mines in Belgium and France. In 1919 he became a partner in the New York firm of Yeatman & Berry. He was a Legion of Honor Member of AIME.

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on December 31, 1953 was 19,718; in addition 2195 Student Associates were enrolled.

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The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

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Morenci—Hunt, Alan W. (R. C/S—S-M)
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Arkansas
Little Rock—Williams, Norman F. (M)

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Berkeley—Hall, Durand A. (R. M)
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Marble—Hill, Arnold E. (R. C/S—S-M)
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Virginia—Martin, John M., Jr. (C/S—A-M)

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Nevada
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Searchlight—Moore, Frank C., Jr. (C/S—A-M)

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Rockaway—Godla, Joseph J. (R. C/S—S-J)

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Miragoane—Ryan, Edwin J. (M)

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Chile, Potrerillos—Mason, Cecil A. (J)
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— Coming Events —

Mar. 10, AIME, Connecticut Local Section, American Brass Co., Torrington, Conn.

Mar. 11, AIME, Utah Section, dinner with Woman's Auxiliary, Salt Lake City.

Mar. 15-19, National Assn. of Corrosion Engineers, Municipal Auditorium, Kansas City.

Mar. 17, AIME, National Open Hearth Steel Committee, Western Section, Rodger Young Auditorium, Los Angeles.

Mar. 18, Central Pennsylvania Coal Producers Assn., Penn-Alto Hotel, Altoona, Pa.

Mar. 23, Techno-Sales Conference, for members, Bituminous Coal Research, Neill House, Columbus, Ohio.

Apr. 5-7, AIME, Blast Furnace, Coke Oven, Raw Materials, and National Open Hearth Conference, Palmer House, Chicago.

Apr. 7, AIME, Chicago Local Section, Chicago Bar Assn., Chicago.

Apr. 21-23, Southern Industrial Wastes Conference, Hotel Shamrock, Houston.

Apr. 24, AIME, Columbia Section, student meeting, Idaho University, Moscow, Idaho.

Apr. 26-28, Canadian Institute of Mining and Metallurgy, Annual Meeting, Mount Royal Hotel, Montreal.

Apr. 26-30, American Society of Tool Engineers' Industrial Exposition, Convention Center, Philadelphia.

Apr. 27, Open Meeting of the Assn. of Consulting Chemists and Chemical Engineers Inc., Hotel Belmont Plaza, New York.

Apr. 29-May 1, Pacific Northwest Metals and Minerals Conference of 1954, joint meeting of Metals Branch and Industrial Minerals Div., Multnomah Hotel, Portland, Ore.

Apr. 30-May 1, AIME, New England Regional Meeting, Bond Hotel, Hartford, Conn.

May 2-6, Electrochemical Society, La Salle Hotel, Chicago.

May 3-5, Coal Convention of the American Mining Congress, Cincinnati.

May 3-8, International Conference on Complete Gasification of Coal, Inichar, Liége, Belgium.

May 8-14, American Foundrymen's Society, Cleveland Auditorium, Cleveland.

May 9-11, Southeastern Retail Coal Assn., Annual Convention, Atlanta-Biltmore Hotel, Atlanta, Ga.

May 17-19, Indiana Coal Merchants Assn., Annual Meeting, French Lick, Ind.

May 24-25, Ohio Coal Conference, Annual Convention, Neill House, Columbus, Ohio.

May 24-27, Symposium on Instrumentation Industrial Hygiene, University of Michigan, Ann Arbor.

May 26-28, Southern Appalachian Industrial Exhibit, Bluefield, W. Va.

May 27, Central Pennsylvania Coal Producers Assn., Penn-Alto Hotel, Altoona, Pa.

June 4, Big Sandy-Elkhorn Coal Operators Assn., Annual Meeting, Lexington, Ky.

June 14-18, Second U. S. Congress of Theo. & Appld. Mech. Meeting, University of Michigan, Ann Arbor.

June 20-24, American Society of Mechanical Engineers, Wm. Penn Hotel, Pittsburgh.

June 23-26, American Coal Sales Assn., Annual Convention, Colorado Springs, Colo.

June 25-26, 1954 Pacific Coast Regional Conference on Clays and Clay Technology, University of California, Berkeley, Calif.

Aug. 2-12, 3rd Convention, Pan American Federation of Engineering Societies, Sao Paulo, Brazil.

Sept. 20-24, American Mining Congress, Civic Auditorium, San Francisco.

Sept. 24, AIME, Minerals Beneficiation Div., Fall Meeting, San Francisco.

Oct. 8-9, AIME Industrial Minerals Div., Fall Meeting, Whiteface Inn, Lake Placid, N. Y.

Oct. 28-29, AIME, ASME Fuels Conference, William Penn Hotel, Pittsburgh.

Oct. 29-30, AIME, Industrial Minerals Div., Rocky Mountain Region Industrial Minerals Conference, Salt Lake.

Nov. 13-18, ASME, 75th Anniversary Meeting, Congress & Hilton, Chicago.

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